OPEN-PIT COPPER MINING METHODS AND PRACTICES, COPPER CITIES DIVISION, MIAMI COPPER CO., GILA COUNTY, ARIZ.

By W. R. Hardwick and M. M. Stover

* * * * * * * * * * * * information circular 7985

UNITED STATES DEPARTMENT OF THE INTERIOR
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BUREAU OF MINES
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OPEN-PIT COPPER MINING METHODS AND PRACTICES, COPPER CITIES DIVISION, MIAMI COPPER COMPANY, GILA COUNTY, ARIZ. [1]

by

W. R. Hardwick[2] and M. M. Stover[3]

SUMMARY

This paper describes open-pit mining methods and practices at Copper Cities Division of Miami Copper Co. in Gila County, Ariz. It is one of a series being prepared by the Burea of Mines on mining methods, practices, and costs in various mining districts of the United States. A similar program, conducted from 1928 to 1939, provided technological information on the development and conservation of our mineral resources and assisted operators and engineers in evaluating and developing new mineral deposits.

This circular contains a brief history of the district and describes the Copper Cities deposit, with particular emphasis on a detailed description of those characteristics--physical, economic, or engineering--that affect the choice of mining method. Prospecting and exploration, sampling and estimation of tonnage and values, development, and mining (including breaking ground, loading, hauling, ore control, and auxiliary operations related to mining) are described. Percentage of extraction, production rate, safety, first aid, fire prevention, mine organization, and processing plant are discussed briefly. The final section gives such costs, on a percentage basis, as are available for publication. Where relevant, some information on the Castle Dome mine is included.

INTRODUCTION

The Copper Cities mine (figs. 1 and 2) is on the south slope of Sleeping Beauty Mountain in sec. 12, T. 1 N., R. 14 E., Gila and Salt River base and meridian, in Gila County, and about 3-1/2 miles north of Miami, Ariz. This district in central Arizona is known as the Globe-Miami District and was the largest producer of copper in Arizona until 1943 when the Morenci mine started open-pit production in the Copper Mountain district of Greenlee County.

Other active copper mines in the district are Miami Copper Co.'s underground mine and Inspiration Consolidated Copper Co.'s Live Oak and Thornton

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1/ Work on manuscript completed June 1958.
2/ Mining engineer, Bureau of Mines, Tucson, Ariz.
3/ Mine superintendent, Copper Cities mine, Miami, Ariz.

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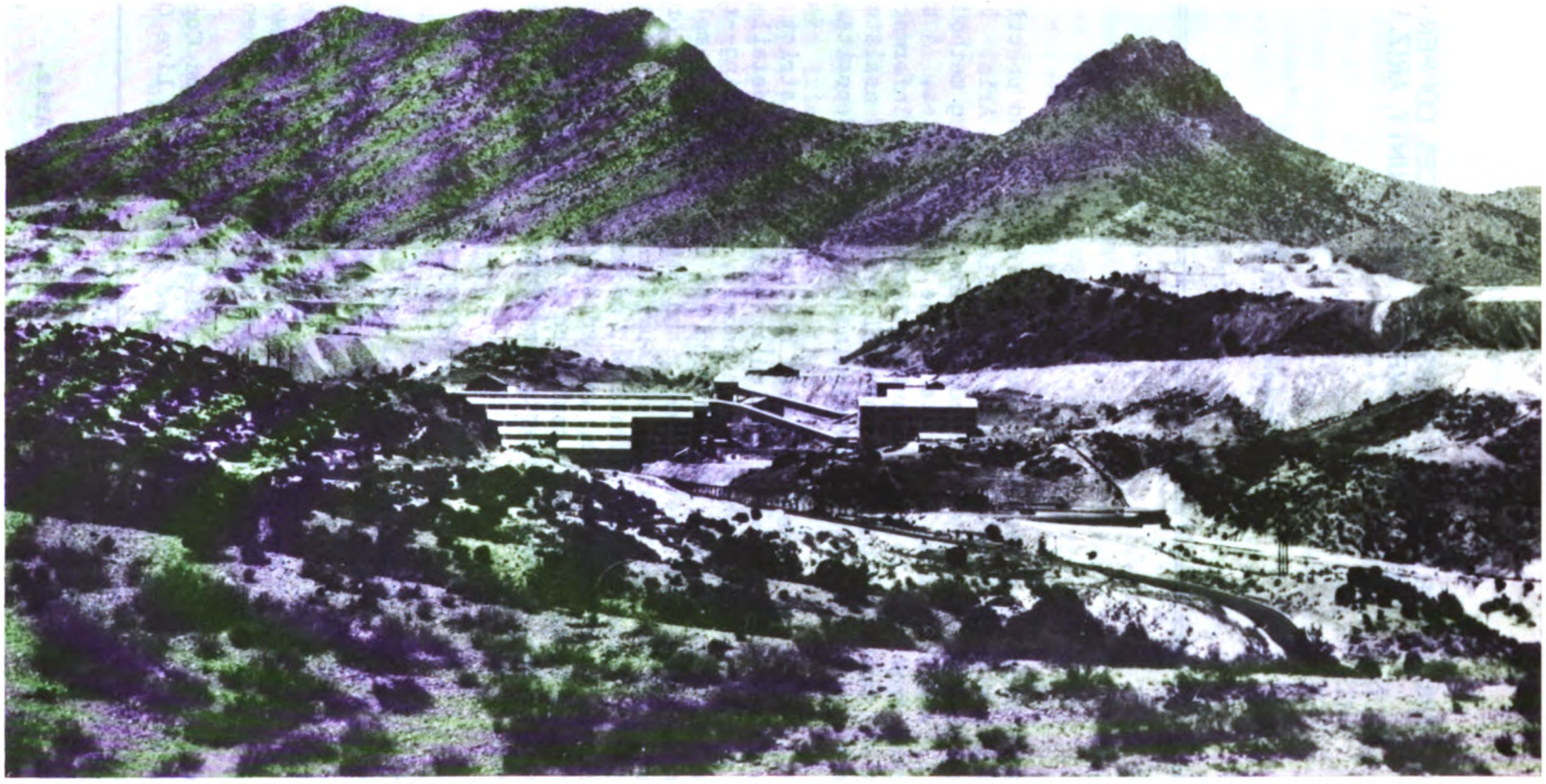

FIGURE 1. - Copper Cities Mine.

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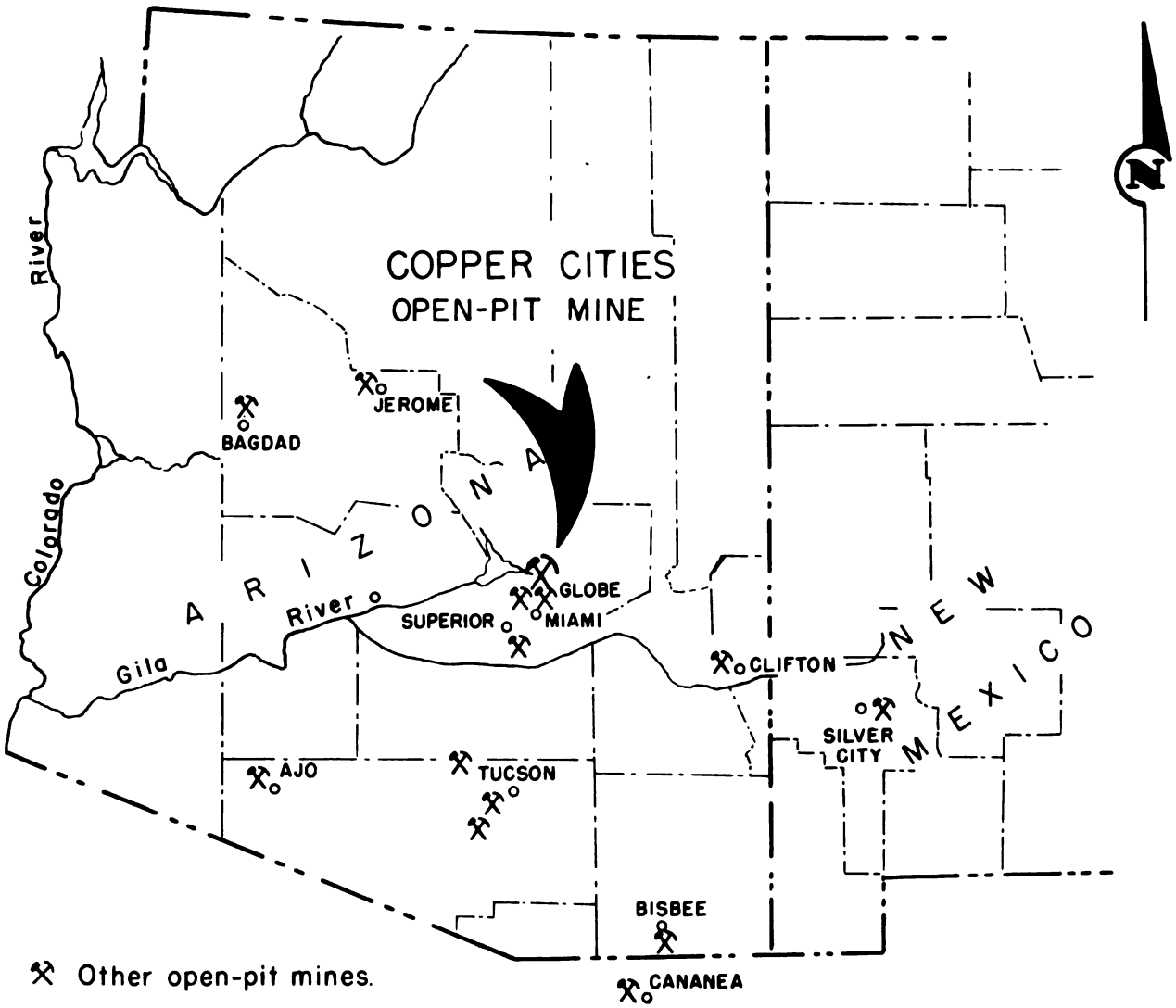

FIGURE 2. - Location of Copper Cities Open-Pit Copper Mine.

open-pit mines. The Old Dominion underground mine and the Castle Dome open-pit mine no longer produce copper ore, but a small amount of copper is produced by leaching the dumps of the Castle Dome mine.

The Copper Cities property was acquired by Miami Copper Co. in 1941. Systematic exploration by churn drill was started in 1943, but operations were hampered by lack of manpower, and activities were halted until the end of World War II. Drilling was resumed in 1946, the exploration program was completed by 1948, development was begun in 1950, and the mine began producing in 1954.

The Copper Cities mine was visited in February 1958, and information for this paper was obtained at that time.

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HISTORY

Some prospecting in the area was done as early as 1901. The first serious attempt at exploration of the Copper Cities area was for gold in 1904. Some low-grade gold ore was mined from the oxidized zone in narrow veins, and a 10-stamp mill was built on adjoining property, but the project failed as a gold mine.

Charles E. Hart, mining engineer, examined the property in 1916 and stated in his report that the west side was in "copper porphyry," with good indications of copper sulfides at 500- to 600-foot depths. A two compartment shaft, 362 feet deep, and 12 churn-drill holes, totaling 9,000 feet, were put down and copper ore was discovered, but the property was not developed at that time.

Miami Copper Co. became interested in the Castle Dome group of mining claims in 1940 and purchased the claims in 1941 after a study of the surface geology and preliminary exploration indicated a considerable ore body underlying the area. Subsequent drilling indicated an ore body containing 30 million tons of ore at Castle Dome and an additional 6 million tons at the adjoining Red Hill deposit. Spurred by the demand for copper during World War II, Miami Copper Co. negotiated a contract with Defense Plant Corp. to finance the project and employed W. A. Bechtel Co. to strip the overburden and construct a concentrating plant. The plant was completed and operation was begun on June 10, 1943. When the Castle Dome plant closed on December 5, 1953, a total of 48,484,188 tons of waste had been removed and 41,442,617 tons of ore had been concentrated, with recovery of about 12 pounds of copper per ton.

The Sleeping Beauty or Porphyry Reserve group of mining claims, which later became the location of the Copper Cities mine, also was consolidated with Miami Copper Co.'s holdings in 1941. A drilling project was begun in 1943 and was completed in 1948. About 100 holes were drilled, and calculations, based on information from these holes indicated ore reserves of 33,800,000 tons with an average grade of 0.7 percent copper. Overburden was estimated to be 34,800,000 tons, of which 20,000,000 tons had to be moved before the mine could begin production. An additional reserve of 9,500,000 tons of slightly lower grade ore was discovered by drilling 9 more holes during 1956.

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Since it was known that the Castle Dome ore body would be exhausted by 1953, Miami Copper Co. decided to synchronize the development stages of the new Copper Cities mine with the final depletion stages of the Castle Dome mine. Stripping of overburden was started on the Copper Cities ore body in November 1950. After the milling operation at Castle Dome mine was terminated, the concentrator was moved to the Copper Cities mine, where it began operating on August 2, 1954.

DESCRIPTION OF THE DEPOSIT

The geology of the Copper Cities mine has been described by Peterson,[4] and the geology of the district has been described by Ransome.[5] The ore body is covered by a weathered outcrop of quartz monzonite and granite porphyry that attracted some attention in 1901. The Copper Cities ore body consists of a blanket of chalcocite ore lying beneath a leached outcrop and above a protore that resulted from primary sulfide mineralization.

The oldest rock formation exposed in the Copper Cities area is the lower Pinal schist of Precambrian age. This was intruded by large masses of granite and, after a period of erosion was covered by the Precambrian Apache group, Cambrian Troy quartzite, Devonian Martin limestone, Mississipian Excabrosa limestone.[6] Then quartz monzonite, diabase, and granite porphyry intruded the older formations, and after a second period of erosion the area was covered by dacite flows and the Gila conglomerate. Recent erosion has removed the Gila conglomerate and much of the dacite in the immediate area of the ore body.

Sometime after intrusion of the granite porphyry and the Lost Gulch quartz monzonite, copper mineralization occurred, and hypogene minerals (quartz, pyrite, chalcopyrite, and molybdenite) were deposited. The approximate outline of the outcrop of mineralized rock covers an area 10,000 feet long by 3,000 feet wide, and the ore body is roughly 1,800 feet square (fig. 3).

The enriched metallized area that forms the ore body is bounded on the north by Sleeping Beauty fault, which strikes about N. 60° E., on the west by the Coronado fault zone, and on the east by the Drummond fault. The thickest and highest grade ore is in quartz monzonite; the ore in granite porphyry is thin. The protore varies from slightly below to well above cutoff grade. Supergene enrichment was important, and the limits of the economic ore body approximate those of the chalcocite zone.

The leached capping ranges from 20 to 115 feet in thickness; the average thickness is 65 feet. The copper content of the cap rock is generally less than 0.1 percent. Malachite, azurite, and turquoise are the most common

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4/ Peterson, N. P., Copper Cities Deposit, Globe-Miami District, Ariz.: Econ. Geol., vol. 49, No. 4, June-July 1954, p. 362.

5/ Ransome, F. L., The Copper Deposits of Ray and Miami, Ariz.: Geol. Survey Prof. Paper 115, 1919, 192 pp.

6/ Work cited in footnote 5.

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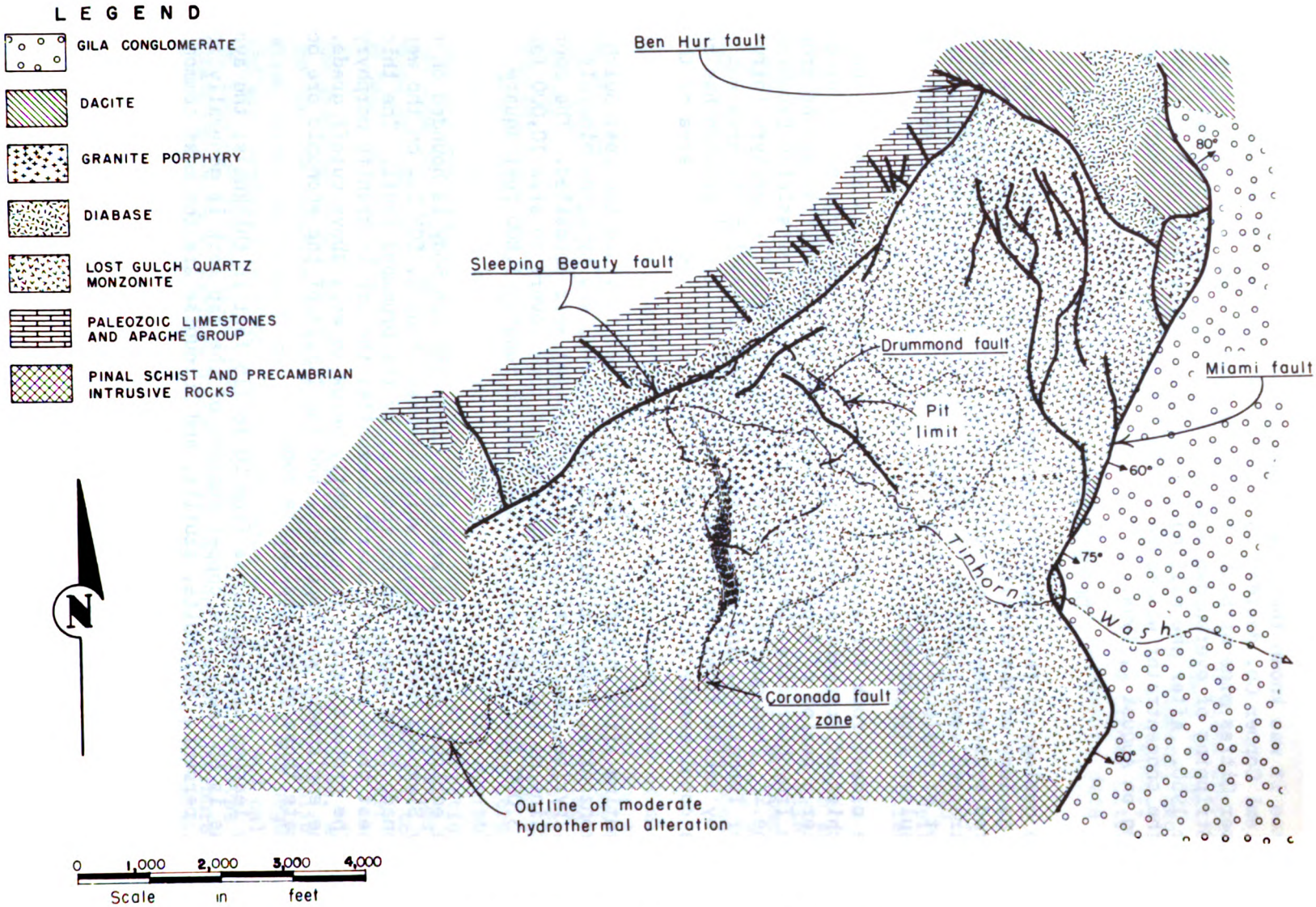

FIGURE 3. - Geology of Copper Cities Mine.

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copper minerals in the capping. Chalcocite enrichment extends from 100 to 220 feet below the leached capping. Most of the ore body contains less than 0.1 percent acid-soluble copper. The transition from oxidized capping to sulfide ore is generally sharp and is marked by an abrupt change in color from light-brown to gray. The level of ground water generally slopes toward the south, following the bed of Tinhorn Wash. The ore body is thick under hills and ridges and thin under gulches. The altitude ranges from 4,250 feet on the highest hill to 3,800 feet in the bed of Tinhorn Wash at the southern limits of the mine.

PROSPECTING AND EXPLORATION

Exploration in the Copper Cities ore body was begun by Miami Copper Co. in 1943 and completed in 1948. The deposit was explored by churn-drill holes that averaged 600 feet in depth. The holes generally were spaced 250 feet apart, forming a rectangular grid, but in questionable areas or near the limits of the ore body they were also drilled at intermediate points. The base grid was oriented so that sections were at right angles to the elongation of the mineralized rock. A map was prepared showing surface geology, topography, and location of drill-hole collars. With the plan map as a base, cross sections and longitudinal sections were made to record assay and geological data of the drill holes.

SAMPLING AND ESTIMATION OF TONNAGE AND VALUES

Sludge samples were taken at 5-foot intervals by passing all the churn-drill cuttings through a splitter that made a 1/64 cut. The sample was reduced to about 15 pounds in a Jones splitter, dried, and assayed. Each 5-foot sample was assayed for total and oxidized copper; composite samples for each 50 feet were assayed for gold, silver, and molybdenum. All samples were examined for rock type and other geological features, and the data were logged.

Sample holes generally were drilled by company crews, although some holes were drilled by contractors. The policy of the company was to sacrifice drilling speed to obtain accurate samples. Holes were collared with a 10-inch bit and usually were finished with a 6-inch bit. All holes were cased from top to bottom, and about 96 percent of the casing was recovered and used in subsequent holes. About 100 holes that averaged 600 feet in depth were drilled to delimit the ore body before 1948; 9 additional holes were drilled in 1956.

All exploration indicated an ore body of relatively small tonnage (43,-300,000 tons) and low copper content. Furthermore, the surrounding area and margins of the deposit had been adequately prospected, and it was considered improbable that mining or additional exploration would disclose enough additional ore or that market conditions would change to justify enlarging the initial pit limits. Therefore, it was of prime importance that initial plans for the pit be the best, economically, for the life of the mine. This meant that pit limits must be planned that would meet a set of conditions to give an

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acceptable minimum profit and yet be extended to the theoretical slope lines for each level as it was completed.[7]

Elevation of the bottom level of the pit was set at 3,600 feet. This level was arbitrary and depended on inspection, cut-and-try, and the judgment of the planners. Selection was influenced by such factors as maximum ore production, working room, pit drainage, and haulage.

Exploration had indicated that cap rock was relatively uniform in thickness and that copper mineralization was gradational. The pit limits were to be assay boundaries rather than geological features, and it was decided that they could be set by consideration of a line rather than a volume, thus reducing the calculations required.

Before acceptable theoretical slope lines were established, certain conditions were assumed. On the basis of experience at Castle Dome mine, it was decided that benches 45 feet high would be satisfactory and that a back slope of 45° could be safely maintained.

The cost to concentrate a ton of ore was assumed to include the mining, milling, and miscellaneous costs per ton. The cost per pound of copper was assumed to include the smelting and miscellaneous costs and the mininum acceptable profit per pound of copper. The mine grade of ore that would satisfy the assumed cost per ton of ore and per pound of copper was calculated by the following formulas:

$$A.\quad \frac{\text{Total cost to mine and concentrate 1 ton ore}}{\text{Gross value per pound copper}} = \text{Net pounds of copper per ton ore to give minimum acceptable profit.}$$

$$B.\quad \frac{\text{Net pounds copper per ton ore}}{\text{Percent extraction}} = \text{Gross pounds copper per ton ore to give minimum acceptable profit.}$$

$$C.\quad \frac{\text{Gross pounds copper per ton ore} \times 100}{2{,}000} = \text{Percent copper required in mill heads to give minimum acceptable profit.}$$

Certain figures are assumed as follows to illustrate these formulas:[8]

Cost to concentrate 1 ton of ore.....	$0.940
Cost to mine 1 ton of waste.....	$0.200
Cost to put 1 pound of copper from concentrate into a market product.....	$0.058

7/ Much of the discussion on the determination of ultimate pit limits is based on a report, "Calculation of Pit Limits and Ore Reserves, Copper Cities Mining Co.", by Gray, J. H., and Simmons, W. W., prepared for AIME, Arizona section, field trip, 1955.

8/ Costs used to illustrate this example are not necessarily those used at Copper Cities mine.

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Market price of 1 pound of copper.....	$0.180
Gross value per pound at minimum acceptable profit.....	$0.123
Percent extraction.....	$0.875

Substitutions in the formulas are as follows:

A. $\frac{\$0.940}{\$0.123}$ = 7.642, which equals the net pounds of copper per ton of ore to give minimum acceptable profit.

B. $\frac{7.642}{0.875(\text{extraction})}$ = 8.734 is equal to gross pounds of copper per ton to give minimum acceptable profit.

C. $\frac{8.734 \times 100}{2{,}000}$ = 0.437 percent, which is the mine grade necessary to satisfy the above assumptions.

The mine grade of ore necessary to meet the above conditions plus the mining of 1 ton of waste is calculated in the same way but must also include the cost of mining a ton of waste. Substitutions in the formulas give the following:

$$\frac{\frac{0.94 + 0.20}{0.123}}{\frac{0.875}{2{,}000}} \times 100 = 0.530 \text{ percent, or } 10.600 \text{ pounds of copper per ton.}$$

The grade necessary to give the minimum acceptable profit with a different waste factor is calculated in the same way, and a table of waste to ore ratios for which a corresponding mine grade of ore that will meet the assumed conditions including profit is presented (table 1).

TABLE 1. - Ore-waste ratio and its required mine grade

Ratio of waste to ore	Mine-grade ore, percent Cu	Ratio of waste to ore	Mine-grade ore, percent Cu	Ratio of waste to ore	Mine-grade ore, percent Cu
0.0:1	0.438	1.7:1	0.598	3.4:1	0.756
.1:1	.447	1.8:1	.606	3.5:1	.765
.2:1	.457	1.9:1	.616	3.6:1	.774
.3:1	.466	2.0:1	.625	3.7:1	.784
.4:1	.475	2.1:1	.634	3.8:1	.793
.5:1	.485	2.2:1	.644	3.9:1	.803
.6:1	.494	2.3:1	.653	4.0:1	.812
.7:1	.504	2.4:1	.662	4.1:1	.821
.8:1	.513	2.5:1	.672	4.2:1	.831
.9:1	.522	2.6:1	.681	4.3:1	.840
1.0:1	.532	2.7:1	.690	4.5:1	.849
1.1:1	.541	2.8:1	.699	4.5:1	.858
1.2:1	.551	2.9:1	.709	4.6:1	.868
1.3:1	.560	3.0:1	.718	4.7:1	.877
1.4:1	.569	3.1:1	.727	4.8:1	.886
1.5:1	.578	3.2:1	.737	4.9:1	.896
1.6:1	.588	3.3:1	.746	5.0:1	.905

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Next, selecting the elevation of 3,600 feet for the bottom bench, the upper-bench levels are drawn on the sections, and the average grade of the sample holes through each bench is plotted along the hole. The grade value is projected to the midpoint between holes. These sections become the work sheets for the determination of the ultimate pit limits. The first step in establishing the back slope is to draw arbitrarily a line on the section at a slope of 45° and located as near the correct position at one end of the ore body as judgment based on a quick visual inspection will permit (line A on fig. 4). This line then is examined to ascertain that it does comply with the acceptable conditions.

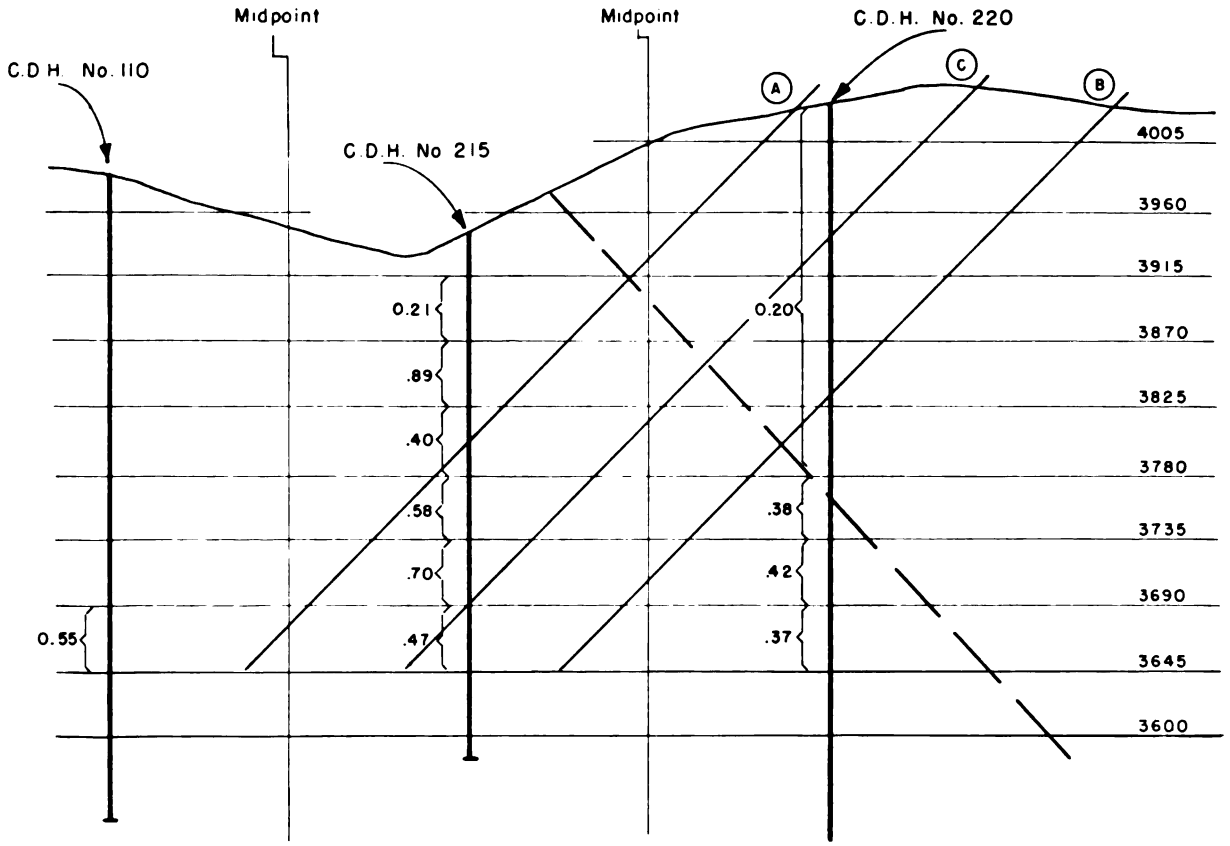

FIGURE 4. - Part of Section N. 3500 Illustrating Investigation of Slope Line.

The total length is measured from the bottom of the lowest ore bench to the point where the line intersects the surface. Next, the integral lengths of the lines along the slope through each ore bench are measured, and each length is multiplied by its respective grade of ore. The sum of these "grade lengths" divided by the sum of the length of slope lines through the ore

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benches is the average grade along the section of the back slope line in ore, and the total length minus the length in ore is the measured length of slope line in waste.

By refering to table 1, a waste-to-ore ratio can be found corresponding to the average grade that has just been computed. A theoretical length of slope line in waste is obtained when this ratio is multiplied by the length of slope line in ore. If this theoretical length of slope line in waste is greater than the length of slope line actually measured, the ultimate slope line lies in a direction away from the pit or ore body. However, if the theoretical length of slope line in waste is less than the measured slope line, the ore along this slope will not pay for the waste, and the ultimate pit limit lies in the direction toward the pit's center. By several attempts, a slope line can be located along which the measured length in waste will equal the theoretical length in waste that can be carried by the measured length in ore under the assumed conditions.

The back slope lines on all sections were calculated by the same process. The detailed calculation of the lines shown on figure 4 are given to illustrate the method.

Calculation of line A (fig. 4) is as follows:

Measured line		Slope line through ore			
Designation	Length, feet	Bench	Length, feet	Grade, percent	Grade length, units
Ore	255.00	3645	45.40	0.55	24.97
Waste	290.40	3645	18.15	.47	8.53
	545.40	3690	63.65	.70	44.56
		3735	63.65	.58	36.92
		3825	63.65	.89	56.65
			254.50	.674	171.63

From table 1: 0.674 percent ore grade will carry 2.5:1 waste to ore; therefore, 254.5 (feet in ore) x 2.5 = 636.25 feet of waste, which 254.5 feet of 0.674 percent ore will carry. As this is greater than the 290.4 feet of waste actually measured along slope line (A), and B was tried.

Calculation of line B (fig. 4) is as follows:

Measured line		Slope line through ore			
Designation	Length, feet	Bench	Length, feet	Grade, percent	Grade length, units
Ore	88.65	3645	63.65	0.47	29.92
Waste	456.35	3690	25.00	.70	17.50
	545.00		88.65	.535	47.42

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From table 1: 0.535 percent ore grade will carry 1:1 waste to ore; therefore, 88.65 feet x 1.0 = 88.65 feet of waste, which 88.65 feet of 0.535 percent ore will carry. As this is less than 456.35 feet of waste measured along slope line B, line C was tried.

Calculation of line C (fig. 4) is as follows:

Measured line		Slope line through ore			
Designation	Length, feet	Bench	Length, feet	Grade, percent	Grade length, units
Ore.....	230.95	3645	63.65	0.47	29.92
Waste.....	329.05	3690	63.65	.70	44.56
	560.00	3735	63.65	.58	36.92
		3825	40.00	1/.50	20.00
			230.95	.569	131 40

1/ Assume 0.50 percent grade from midpoint to fault.

From table 1: 0.569 percent ore grade will carry 1.4:1 waste to ore; therefore, 230.95 feet of 0.569 percent ore will carry 323.33 feet of waste. The slope line through waste actually measures 329.05 feet. As this is as close as the accuracy of data, line C was used as the theoretical slope line.

After the back slope had been established on all sections in this manner, the data were transferred to plan maps. A key level was selected near the midpoint of the ore column and a composite plan map was constructed by mechanical development from the key level up the slope to the surface and down the slope to the pit bottom. Maps for each level then were made by transferring the theoretical bench outlines from the composit map and plotting the geology and the data for the prospect holes with the average grade from the sections. Polygons were drawn to cover the area of grade influence, modified by geology around each drill hole, and the area was determined by planimeter. From these data the average grade and the tonnage of ore and waste on each bench were computed. The tonnage of ore and waste in the pit was obtained by totaling that from all levels. Tonnage from each level was weighted by the average grade of that level to obtain the average grade of the ore body.

This method was used to estimate the tonnage and grade of the Copper Cities ore body, and based on the initial exploration drilling the ore body was estimated to contain 33,800,000 tons of ore, with a grade of about 0.7 percent copper. A cutoff of 0.4 percent copper was used, and the waste that must be moved to mine this ore was estimated at 34,700,000 tons, or an ore-to-waste ratio of 1 to 1.03. Drilling nine holes in 1956 increased these reserves by 9,500,000 tons of ore.

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DEVELOPMENT

After the decision was made to develop the Copper Cities mine, a detailed topographic map was necessary to plan subsequent operations. In 1949, when a shortage of engineers existed, Fairchild Aerial Surveys, Inc., was employed to produce a topographic map of the area on a scale of 1 inch equals 40 feet.9/ All cut-and-fill work for crusher and mill foundations, stripping tonnages, and areas for waste dumps, tailings ponds, and road fills were planned from this map.

It was estimated that 20,000,000 tons of the total waste would have to be moved before the concentrator started operating. The company knew that the Castle Dome ore body would be exhausted in 1953, and after detailed planning, with due regard for the time element, stripping the overburden from the Copper Cities ore body was started in November 1950. By January 1951, two 2-1/2-yard diesel shovels and four 15-ton rear-dump trucks were stripping waste. The 2-1/2-yard shovels and 15-ton trucks were used early in the operation and later to open new benches where room was restricted. By the end of the first 6-month period, a 5-yard electric shovel and eight 30-ton trucks were moved to the new mine. Stripping began at a rate of 200,000 tons per month, increased to 500,000 tons per month, and later to 800,000 tons per month. Three 5-yard shovels were operated a total of 5 shifts per day and 6 days per week. By August 2, 1954, when the 12,000-ton concentrator was ready to start operating, about 19,500,000 tons of waste had been stripped and the mine was ready to deliver the required daily tonnage to the mill.

MINING

The general mine layout (fig. 5) is so arranged that waste is disposed of on each side of the open pit and the ore transported to a 1,000-ton capacity bin at the crusher. The general surface slopes rather steeply to the south along Tinhorn Wash, which cuts through the area over the ore body. The initial levels take the form of crescents advancing north into the higher ground; the lower levels will have a shape more like a horseshoe. Generally, three levels are being operated; one level is approaching completion at the final pit limit, one level is in full production, and one level is just being started. This situation is well illustrated in figure 5, which shows a new level, the 3780, being started.

In general, the plans of operation for stripping and mining are similar. However, they differ in detail, as they did for example, at the start of the stripping operation when all diesel equipment was being used. The details of the mining cycle fall naturally into the steps of breaking, loading, transporting, and disposing of ore and waste. These are discussed in the following sections. Differences between mining and stripping details are discussed where they occur. Major items of mine equipment are summarized.

9/ Castle Dome Becomes Copper Cities in Synchronized Mine and Mill transfer: Min. World, vol. 16, No. 6 May 1954, p. 34.

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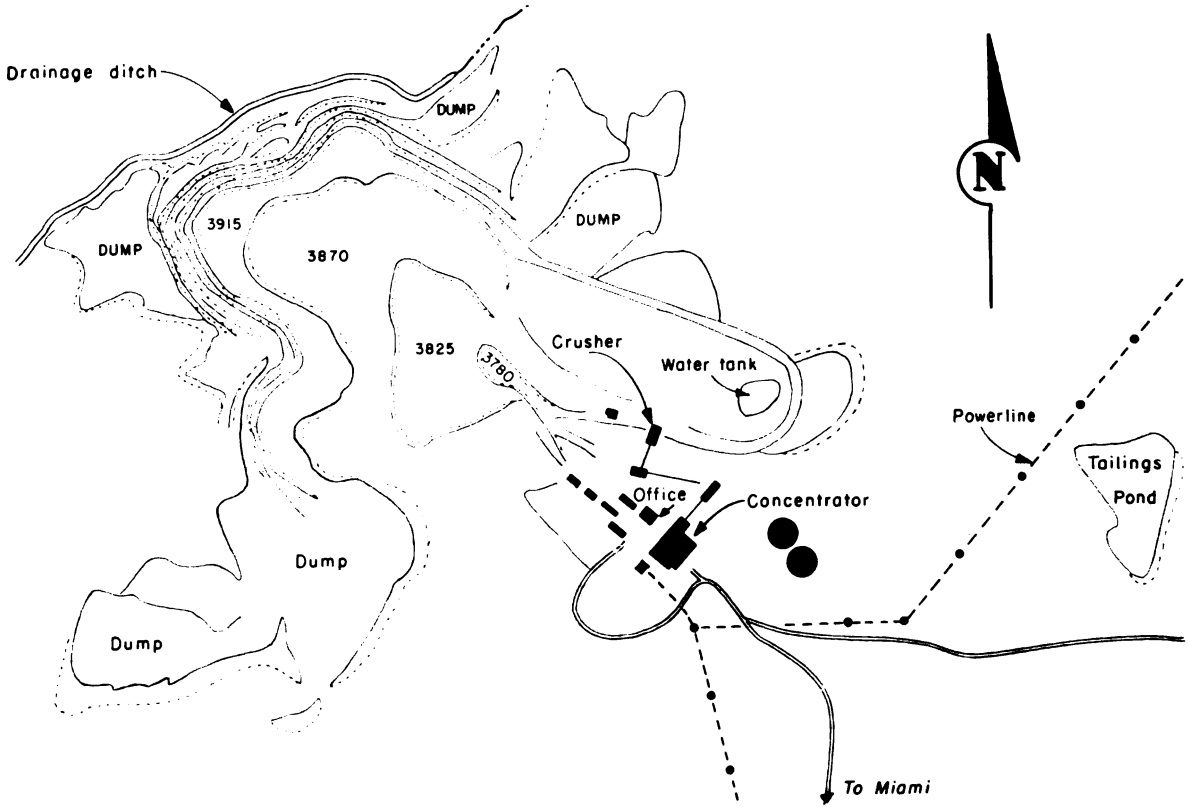

FIGURE 5. - General Arrangement of Mine and Plant, Copper Cities Division.

Mine equipment

	Number of units
Prospecting drills:	
Spudder-type churn drills converted to 1,000-foot capacity..	3
Production drills:	
Medium-sized diesel-powered rotary drill.....	1
Small spudder-type churn drills (used to clean holes).....	3
Loading equipment:	
Full-revolving electric-powered 5-yard shovels.....	3
Haulage equipment:	
40-ton rear-dump trucks.....	8
30-ton rear-dump trucks.....	2
Road equipment:	
Bulldozers.....	7
Grader.....	1

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	Number of units
Miscellaneous trucks:	
2-ton flat rack (supplies)	1
2-ton flat rack (powder)	1
4-yard dump (stemming)	1
3,000-gallon sprinkler	1
Special equipment:	
3 kv.-a. generating plants (dump lights)	2
Sump pumps, 50-gallon per minute	3

Breaking Ground

At the Copper Cities mine, holes with 9-inch diameters are drilled with churn or rotary drills along predetermined lines. Multiple rows of holes are drilled but are blasted one row at a time to break the ore and waste. The ground is easily drilled, and blasting characteristics are generally uniform. Ground water is a problem in some areas.

"Breaking Ground" is accomplished by a drilling and blasting department that employes 11 men, including 2 foremen and a sampler. The process involves the following steps: Primary drilling, primary blasting, and secondary drilling and blasting. These are described in the following sections.

Primary Drilling

At the start of the stripping operation, primary drilling was done with a light gasoline-powered churn drill. The string of tools weighed about 1,500 pounds, and the machine drilled at the rate of about 66 feet per shift. When electric power became available, the gasoline-powered machine was replaced by a heavier churn drill with a 2,700-pound string of tools. This machine drilled 9-inch holes at the rate of about 104 feet per shift. As the pit became deeper, sulfide ore became harder to drill, and the average dropped to about 98 feet per shift.

In 1956 a medium-sized rotary drill was purchased (fig. 6). Powered with a diesel engine, this machine drilled a 9-inch blasthole with a standard roller cone bit at the rate of 461.6 feet per shift. During one shift a record 1,125 feet of blasthole was drilled. In 1958 this machine drilled enough blasthole footage in one shift per day to sustain the current production of 20,000 tons of rock per day.

Three churn drills, one powered by electricity and two by gasoline, are available for emergency use. These drills are also used to clear water from wet holes before they are loaded with powder and to drill deep prospect holes and blastholes in isolated areas where moving the rotary drill would be time consuming.

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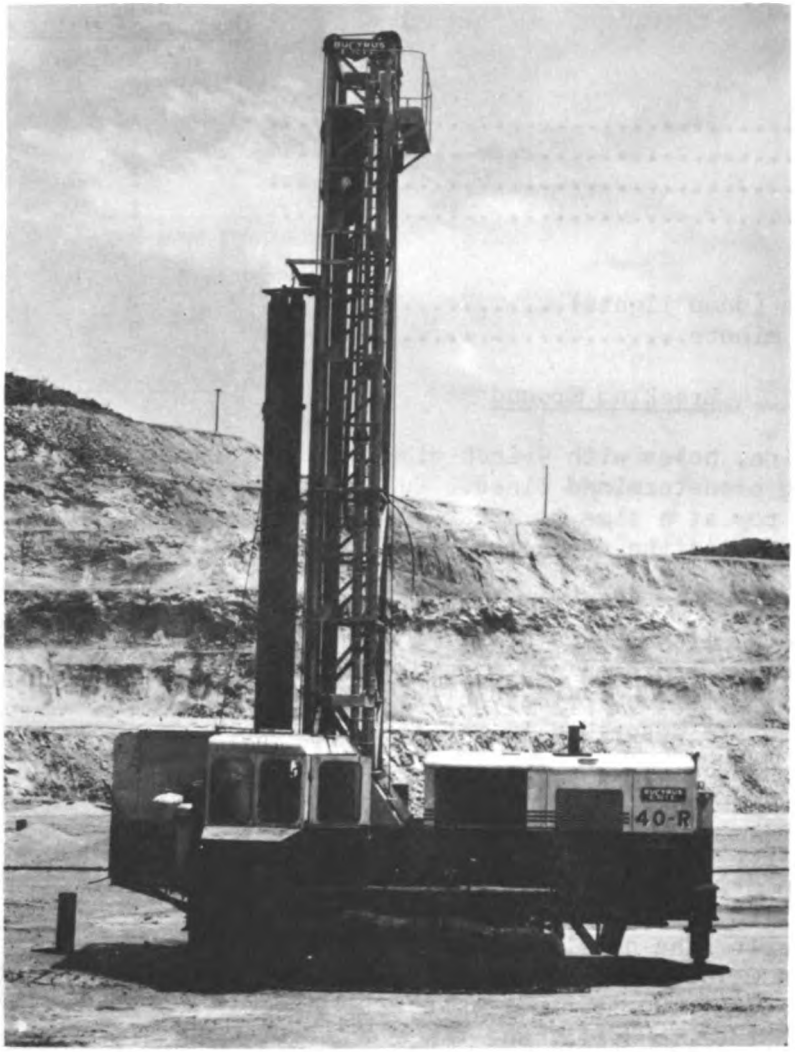

FIGURE 6. - Rotary Drill.

A drill crew consists of a driller and a helper. The drill operator prepares and submits reports for each shift to the drilling and blasting foreman (figs. 7 and 8). An important section of the churn drill report lists the supplies and acessory tools that are needed at the drill for the next operating shift. The reports are filed in the office of the drilling and blasting foreman, and information from them is accumulated to form a permanent drilling and blasting record.

The direction of shovel cuts is determined and planned in advance on a map by the mine superintendent. The general spacing of the holes along a row and the spacing between the rows of holes are planned, and then lines are staked in the field by a transit party from the engineering department. Stakes to mark where the holes are to be drilled are distributed along these lines by the drilling and blasting foreman, who may make minor changes in hole spacing to adjust for local rock conditions. Holes are drilled at intervals of 20 to 30 feet along lines that are 35 to 40 feet from each other (fig. 9). All holes are drilled to 6 feet below shovel grade.

An entire area is completed before the drill is moved to another bench. Short pieces of 9-inch-diameter carboard tube are placed in the holes as the bit is being removed to prevent the cuttings from sloughing back in the hole. Generally, drilling is 3 to 6 months in advance of blasting and mining. Careful operation of the shovel is necessary so that it does not dig out the tops of the next row of holes before they are blasted. This would result in hard bottom. Figure 10 shows that only the front row of holes is loaded, but the pipes in the background indicate the entire area has been drilled.

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ROTARY DRILL SHIFT REPORT

DRILLER _____ LEVEL _____

HELPER _____ DATE _____

HOLE NO.												
TOTAL DEPTH												
FROM DEPTH												
DRILLED TO												
FT. DRILLED											TOTAL FEET DRILLED	
STARTING DRILL TIME												
FINISH HOLE TIME												
DRILLING TIME											TOTAL DRILL TIME	
MOVING TIME											TOTAL MOVING TIME	
DELAY TIME											TOTAL DELAY TIME	
EXPLANATION											REMARKS	
TOTAL TIME												

BIT CHANGES

HOLE NO.	DEPTH AT CHANGE	SER. NO. DULL BIT	SER. NO. NEW BIT	MAKE OF BIT	MODEL OF BIT	REPAIRS NEEDED

OIL IN DIESEL ENGINE	
OIL IN COMPRESSOR	
FUEL USED	
LUBE OIL USED DIESEL	
LUBE OIL USED COMPRESSOR	

FIGURE 7. - Rotary-Drill Shift Report.

Bits are changed as necessary. Rotary bits are discarded when no longer usable. Churn-drill bits are resharpened and tempered and then returned to service. A bit-sharpening shop is maintained, but it is not operated continuously. When churn-drill bits accumulate, a crew of 2 men, selected from the mine crew, sharpens about 25 bits per shift. Bits on the rotary drill last an average of about 1,668 feet and then are removed and discarded. Bit performance is shown in table 2.

TABLE 2. - Drill performance

	Development	1956	1957
Churn drills:			
Gasoline:			
Feet per shift.....	66	-	-
Feet per bit change.....	72	-	-
Electric:			
Feet per shift.....	104	98	-
Feet per bit change.....	108	112	-
Rotary drill:			
Feet per shift.....	-	-	461.6
Feet per bit change.....	-	-	1,668

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COPPER CITIES DIVISION

MIAMI COPPER COMPANY

MINE DEPARTMENT

CHURN DRILL SHIFT REPORT

Drill No...... **Shift**..... **Level**..... **Date**.....

Relieved Driller..... Time..... Relieved by Driller..... Time.....

Hole Number					
Total Depth					
Started Drilling—Time					
From—Depth					
Drilled to—Depth					
Footage Drilled					
Finished Hole—Time					
Ore or Waste					
Drilling Time					
Moving Time					
Delay Time and Explanation					
TOTAL TIME					

BIT CHANGES

HOLE NUMBER	DEPTH AT CHANGE	NO. DULL BIT	SHARP BIT NO.	REMARKS

SUPPLIES	NO. ON HAND	NO. NEEDED
Bits—Sharp		
Bits—Drill		
Casings—Good		
Casings—B. O.		
Jack Blocks		
Jack Bars		
Crosses		
Lubricants		
Water Hose		
Report Sheets		
Casing Drivers		
Power Cable		
Bit Wrenches		
Safety Belt		
Cable Cutter		
Grease Gun		
Water Barrel		
Light Bulbs		

Driller.....

Helper.....

FIGURE 8. - Churn-Drill Shift Report.

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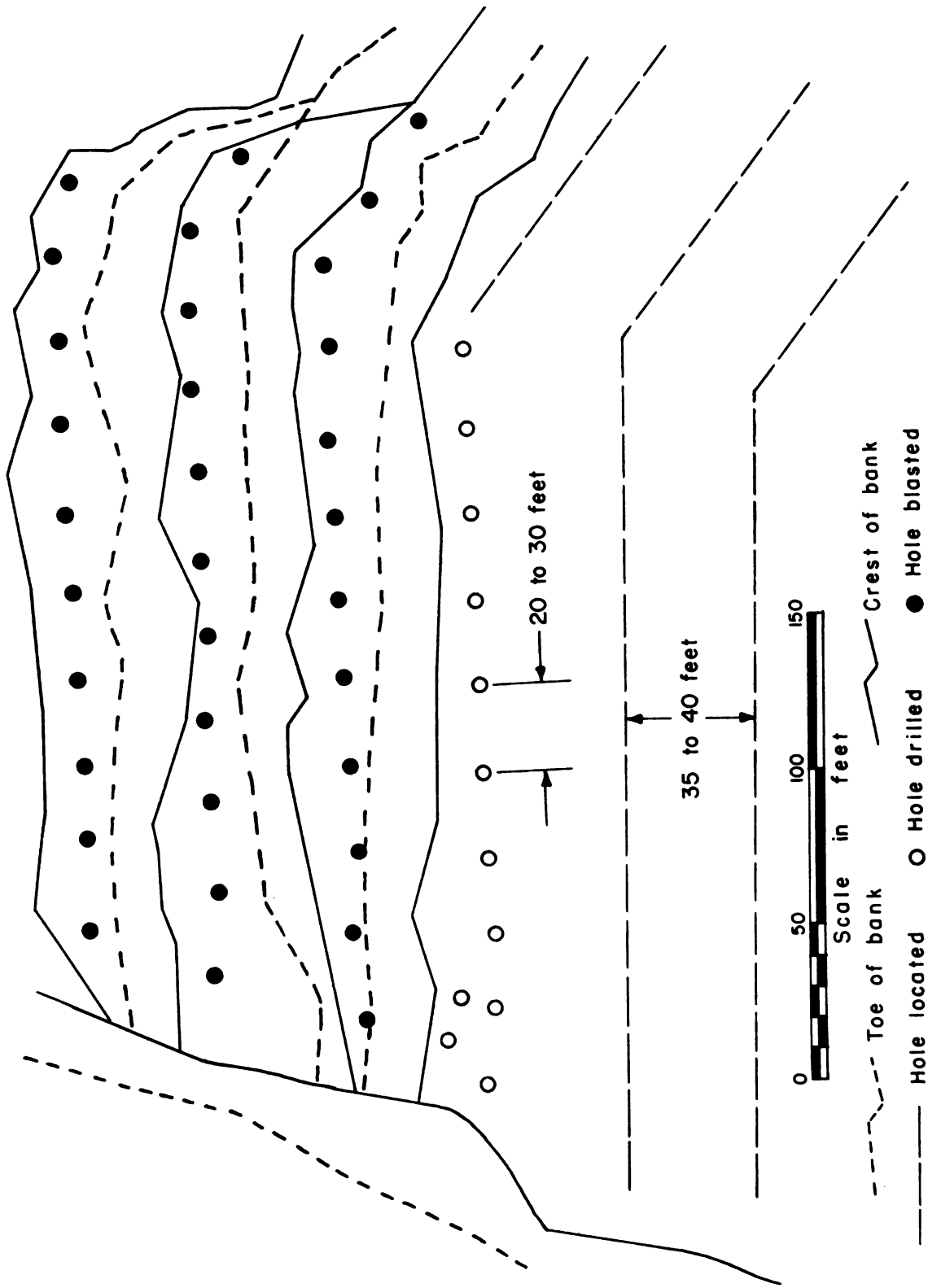

FIGURE 9. - Arrangement of Located, Drilled, and Blasted Holes.

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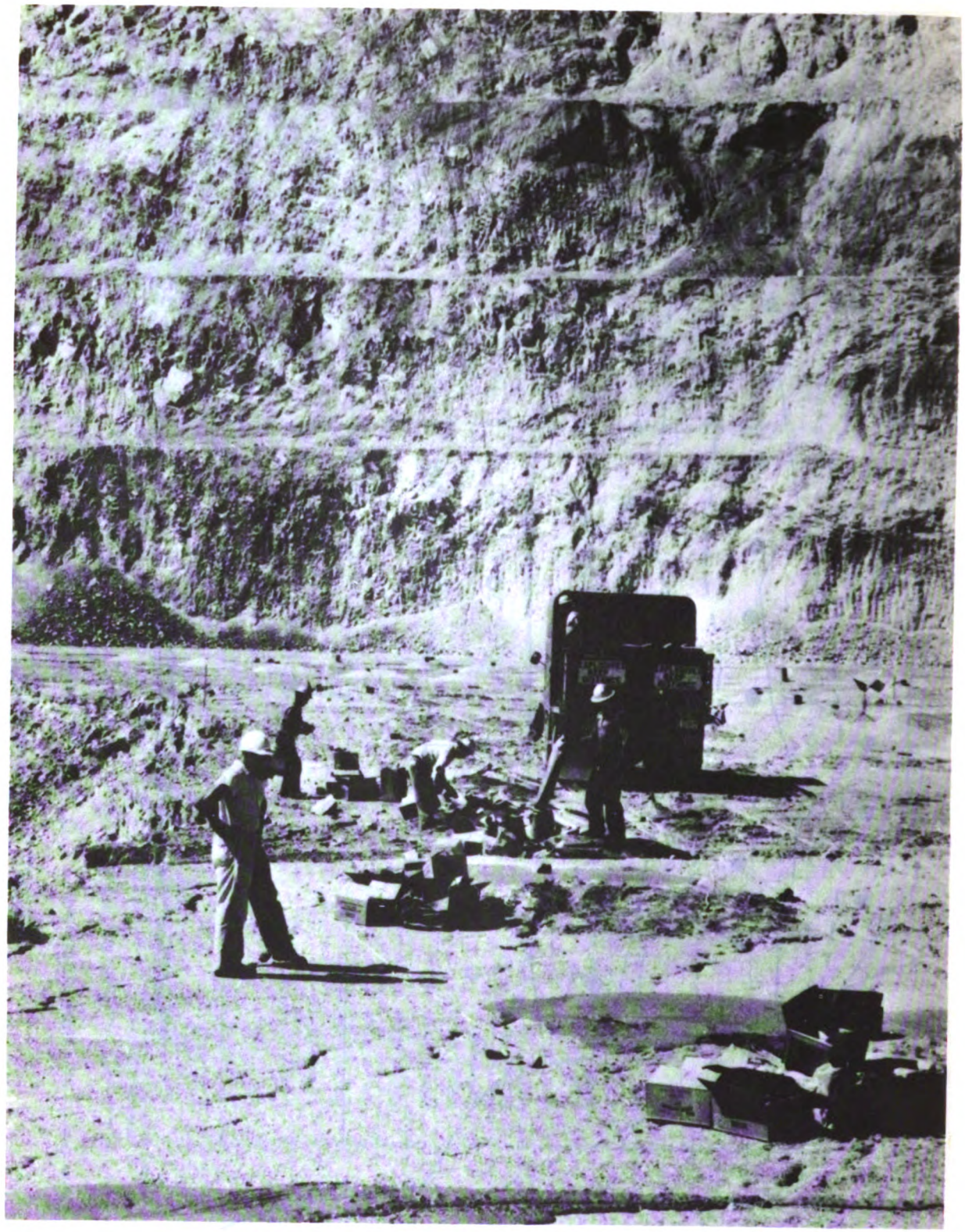

FIGURE 10. - Loading Blastholes.

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After a mining level is completed, a report is made by the mine superintendent, which includes drilling and blasting performance. Some information on a typical level is shown in table 3.

TABLE 3. - Drilling results for a completed level

	Ore	Waste	Total
Tons mined.....	4,332,535.0	6,372,801.0	10,705,336.0
Feet of drill hole blasted.....	51,157.0	75,156.0	126,315.0
Tons mined per foot of hole blasted...	84.7	84.8	84.8
Tons mined per pound of powder used...	7.1	7.1	7.1

Primary Blasting

The primary-blasting crew consisted of three powder loaders, two light-truck drivers, and a powder foreman. This crew loaded and blasted six holes in 2 hours. Primary blasts range from 1 to 15 holes and generally average about 12 holes.

The holes in the area to be blasted are checked by the powder foreman, and if necessary, water or cave material is cleaned from them before loading is begun. The powder foreman checks the depth of the hole, distance to the next hole, and distance from the hole to the crest with a tape; then he estimates the toe distance. All linear measurements are in yards. The average hole-to-toe spacing is computed, and these items are entered on the Field Record of Blast (fig. 11) opposite the hole number. The powder foreman then computes

FIELD RECORD OF BLAST

BENCH _____ DATE _____ WEATHER _____ BOSS _____

BLAST NO. _____ COORDINATES FROM _____ TO _____

HOLE NO.	DEPTH FT.	SPACING-YARDS			AVERAGE HOLE TO TOE	BURDEN CU. YD.	RATIO	POWDER		STEM FT.	DECK LOAD		STEM FT.
		HOLE	CREST	TOE				KIND	BOXES		KIND	BOXES	

FIGURE 11. - Field Record of Blast.

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the charge by referring to a table in which the amount of powder that is required is listed in columns opposite the two variables (fig. 12). The variables, average toe and hole spacing, are listed in the table for intervals that are usually encountered. The body of the table reads in units of boxes of powder (50 pounds). The table is computed for a factor of 0.5 pound per cubic yard. The powder foreman adds or subtracts from the amount indicated in the table, depending on the character of the ground, and enters the amount of powder in the blast record as the powder is loaded in the hole.

AVERAGE TOE IN YARDS											
SPACING IN YD.	4	5	6	7	8	9	10	11	12	13	14
5	4 3	5 4	6 5	6 6	7 7	8 8	9 9	10 9	11 10	11 11	12 12
5 1/2	4 4	5 5	6 6	7 7	8 7	9 8	10 9	11 10	12 11	13 12	14 13
6	5 4	6 5	7 6	8 7	9 8	10 9	11 10	12 11	13 12	14 13	15 14
6 1/2	5 4	6 6	7 7	8 8	9 9	10 10	12 11	13 12	14 13	15 14	16 15
7	5 5	7 6	8 7	9 8	10 10	11 11	13 12	14 13	15 14	16 15	17 17
7 1/2	6 5	7 6	8 8	10 9	11 10	12 11	13 13	15 14	16 15	17 17	18 18
8	6 5	7 7	9 8	10 10	12 11	13 12	14 14	16 15	17 16	18 18	20 19

POWDER IN CASES FOR 45' BENCH
1/2 FACTOR - 6' BELOW GRADE

FIGURE 12. - Powder-Loading Chart.

Before loading the powder, two strands of plastic covered Primacord are tied to a rock and lowered to the bottom of the hole. After the hole is loaded, stemming is placed from the chute on the end of a dump truck (fig. 13). Provision is made to load the truck with fine ore to be used as stemming, on the fine-ore bin. This material has been passed through a 5/16- by 1-inch screen opening. The cuttings from the rotary drill were tried for stemming material but did not give satisfactory results.

Holes are charged with Special Gelatin dynamite in the wet part and Carbamite in the dry part of the hole. The dynamite has a strength of 60 percent and a velocity rating of about 17,400 feet per second. The Carbamite is rated at 70 percent and has a density of about 65 pounds per cubic foot.

The field record of the blast is completed as the holes are loaded and is turned in to the drilling and blasting foreman after the blast is fired (fig. 14), usually at the end of the shift. Information is passed to the engineering department, and a Blast Report (fig. 15) is completed. Each blast

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FIGURE 13. - Stemming a Blasthole.

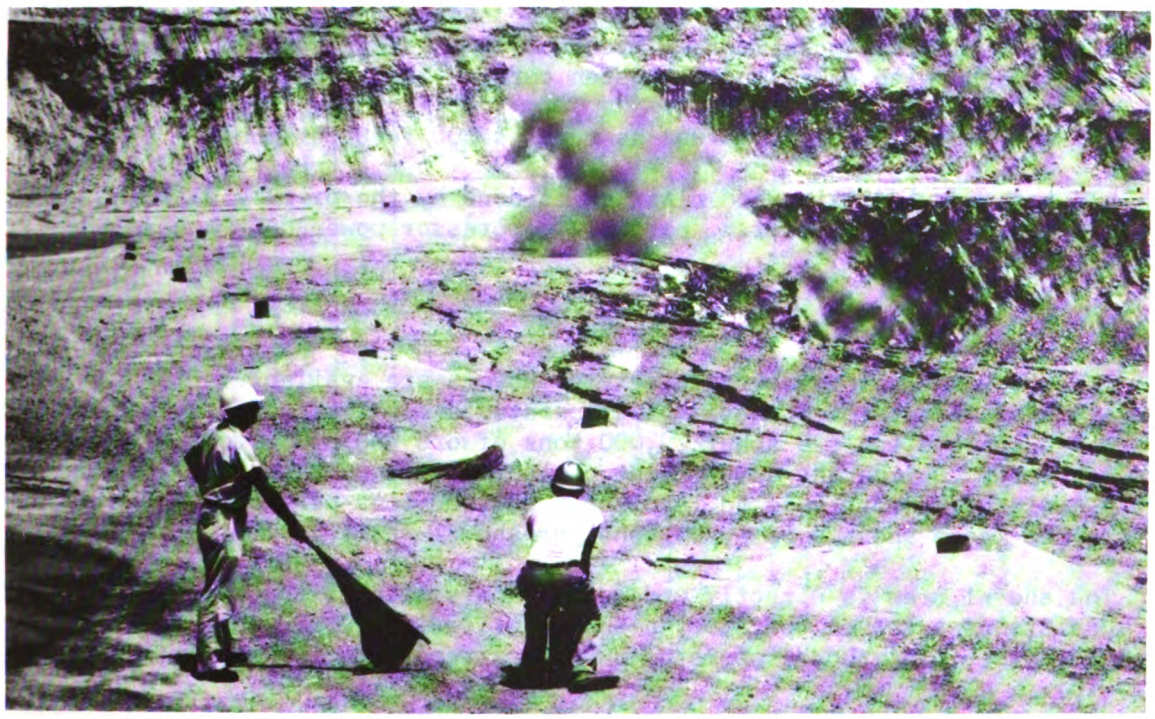

FIGURE 14. - Firing a Primary Blast.

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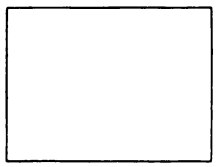
BLAST REPORT



BLAST NUMBER_____
LEVEL_____
CLASSIFICATION_____
DATE BLASTED_____
DATE COMPLETED_____

	ORE	WASTE	TOTAL
ESTIMATED TONNAGE			
TONS MOVED			
REMAINDER			
TONS MOVED			
REMAINDER			

REMARKS_____

STATION	PLANIMETER READING	FACTOR	AREA	DIST.	CU. FT.

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Form 322 Jahn-Tyler

COPPER CITIES DIVISION
MIAMI COPPER COMPANY
MINE DEPARTMENT

DAILY SHOVEL REPORT

SHOVEL NO._____ SHIFT_____ BENCH_____ DATE_____

LOADS					DELAYS						EXPLANATION OF DELAYS
TRUCK No.	KNUCKEY LOADS		EUCLID LOADS		DRILLING AND BLASTING	MECHANICAL	ELECTRICAL	WAITING ON TRUCKS	MOVING	OTHER	OVER 5 MINUTES—GIVE DETAILS LESS THAN 5 MINUTES—CHARGE TO LOADING TIME
	ORE	WASTE	ORE	WASTE							

CONDITION AT BEGINNING OF SHIFT
- PIT AND GRADE_____
- SHOVEL_____
- WATER LINE_____
- POWER CABLE_____

GALLONS FUEL OIL TAKEN DURING SHIFT_____

MOVED TO SHOVEL No._____ AT_____ M. REASON:

PITMEN_____

_____

OILER_____

OPERATOR_____

RELIEVED AT_____ M.

FIGURE 16. - Daily Shovel Report.

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The shovel pit, the entire working area, and level are kept clean at all times (fig. 17). All spill is pushed into the bank where it can be loaded, boulders are broken and loaded, and when humps above digging grade result from hard bottom they are drilled, blasted and the excess pushed into the bank for the next shovel cut.

FIGURE 17. - Loading Operation.

Trucks may be loaded from either side of the shovel (fig. 18). Power is carried to the shovel by an insulated cable. Because heavy trucks cannot drive over the cable without damaging it, the cable is carried over the road on one side of the shovel by a fabricated overhead support (fig. 19). This support can be engaged by a bracket on the canopy of a truck and raised when the dump body is elevated. The truck then can move the cable support to a new position (fig. 20).

Haulage

Transportation problems, similar to those experienced during the early part (prior to 1948) of the mine operation at Castle Dome were encountered at the Copper Cities mine. The ore haul was generally level or downhill. In 1943 when the Castle Dome mine was developed, off-highway trucks had not been as highly perfected as in 1954, when the Copper Cities mine went into full production. After 1948, the lower levels of Castle Dome and the Red Hill ore bodies were mined and long hauls on adverse grades were required. During the

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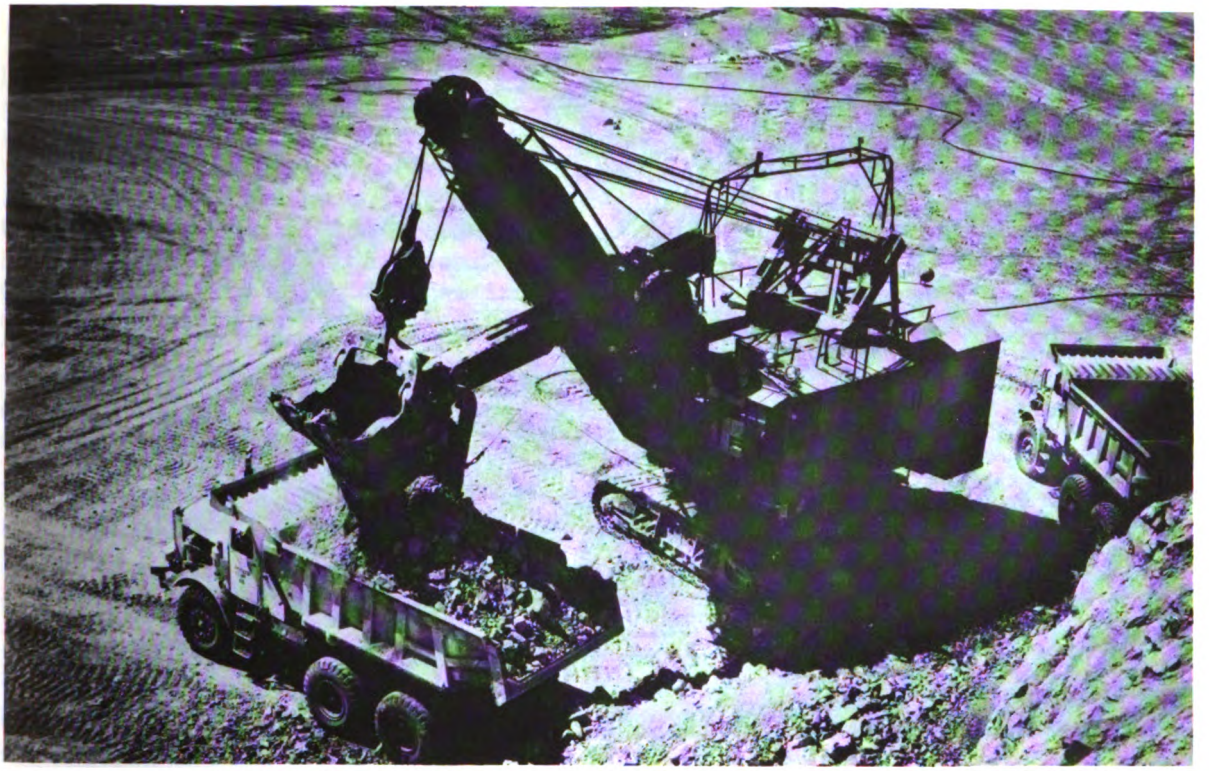

FIGURE 18. - Two Trucks in Loading Position by a Shovel.

the latter part of 1948, a short haul of 3 percent plus became necessary, and the grade became progressively steeper.

The trucks used in 1951 were equipped with 22-yard (or 30-ton) bodies, manually operated transmissions, double reduction differentials, four-stage telescopic hoists, air brakes, and chain drive from jackshafts to each rear axle. From 1943 to 1948, these trucks were powered with a 150-hp. diesel engine. However, after 1948, this engine was replaced with a 275-hp. supercharged diesel, and heavier clutch, transmission, and differential were installed.

During a 3-month period in early 1951, these trucks hauled 933,113 tons of ore an average distance of 0.975 mile, with an average vertical lift of 162.9 feet.[10] All haul roads included 600 to 3,000 feet of plus 8 percent grade. The operating efficiencies were as follows:

Trip per shift	31.7	
Tons per trip	32.9	
Tons per shift	1,045.0	
Average haul, miles	1.8	(Round trip)
Ton miles per shift	941.3	

10/ Van De Water, J. C., Truck operation at Castle Dome: Pres. Arizona AIME, Miami, Ariz., May 7, 1951, 4 pp.

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FIGURE 19. - Overhead Power-Cable Support.

FIGURE 20. - Truck Moving a Power-Cable Support.

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The same trucks that were used at Castle Dome mine were transferred to Copper Cities mine and used during the initial development stages. As these trucks became older, it was apparent that at some point they would have to be replaced with new equipment. A formula has been devised by Miami Copper Co. to assist in the determination of the point of uneconomical operation for pit equipment. New trucks must be paid for over a given period of time, which can be either the life of the truck or the life of the mine, whichever is shorter. At the Copper Cities mine, Miami Copper Co. has elected to pay for a truck during the period while the new truck has minimum maintenance. The period of minimum maintenance is set at 3 years or about 15,000 operating hours.

When the base period for taking capital investment into the operating costs is established at 3 years, the point of uneconomical operation is determined as follows:[11]

Let a = Amortization per cost unit,

m = Average maintenance cost per cost unit during the base period used for amortization, corrected to the current period,

M = Current maintenance per cost unit.

Then, when M = m + a, buying a new truck would be trading dollars. When M is greater than m + a, buying a new truck would show a saving.

Using the 30-ton truck described, the following example illustrates the application of the formula:

Base period maintenance costs:

Average from cost sheets, 1948-50:

	Per ton hauled
Maintenance labor	$0.01295
Truck parts, etc	.01861
Total	.03156
Correct base period costs to 1954 conditions:	
Increase labor by 28.46 percent	.01664
Increase parts cost by 16 percent	.02159
m =	.03823
Amortization of similar new truck in 3 years	.01502
m + a =	.05325
First 6 months, 1954, costs:	
Maintenance labor	.01506
Truck parts, etc	.02188
M =	.03694

11/ Coil, B. R., What is the Economical Point of Replacement of Pit Equipment in the Southwest: Pres. AIME meeting, Miami, Ariz., May 7, 1951.

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Since M is less than m + a, the conclusion is that the point of uneconomical operation has not been reached and new trucks are not justified. This formula applies to the replacement of identical units. Purchases of new equiptment to increase capacity or replacement because of obsolescence resulting from technological improvements must be considered separately.

At Copper Cities mine, four 15-ton trucks were used with 2-1/2-yard diesel shovels in the preliminary stripping, opening new benches, and road building jobs, where their shorter turning radius could be used to advantage. As soon as benches and regular hauls were established; 5-yard shovels and 30-ton trucks were used. In 1953 these 30-ton trucks gave the following performance:

Trips per shift	29.96	
Tons per trip	30.97	
Tons per shift	927.80	
Average haul, mile	0.97	(Round trip)
Ton miles per shift	452.00	

New trucks were purchased in 1956. These were 40-ton (26 cubic yards struck) rear-dump trucks powered with two 250-hp. diesel engines placed side by side. Power is transmitted by torque converters mounted integrally with engines through hydraulically activated multiple disk clutches and transmissions with planetary gearing. Both rear axles are full floating with planetary reduction gears in each wheel. Total reduction is 16.07:1. The frame is rigid and is supported on springs. Two-shoe internal expanding brakes are actuated by compressed air, and the load is dumped by three-stage double-acting hydraulic hoists. The dumping angle is 67°. The net weight of the truck is 78,000 pounds, and the payload is 80,000 pounds. Some general dimensions are presented in table 4.

TABLE 4. - General dimensions of 40-ton truck

	Feet	Inches
Wheel-base	15	0.0
Cab guard to ground	11	11.0
Loading height	11	.5
Chassis length	27	7.5
Overall length	32	5.0
Overall width	12	4.5
Front-axle track	9	10.5
Drive-axle track	8	7.5
Front-axle clearance	1	5.5
Body length	18	11.5
Body width rear	9	11.0
Body depth	4	4.0
Outside front-tire spacing	11	9.0
Outside drive-tire spacing	12	4.5
Turning diameter on outside front tire	62	6.0

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When the mine was visited in February 1957, there were seven of these trucks available, and they were being used almost exclusively. Six trucks were operated during each shift, and their availability was about 91.5 percent. Truck-shifts range from 5.3 to 5.9 and average about 5.47 per shovel-shift. Truck performance for the shift is reported by each driver (fig. 21). The driver also checks and signs a fuel charge slip when his truck is serviced (fig. 22). During 1957 these trucks hauled over roads that were level or slightly in favor of the load with the following performance:

Trips per shift	30.08	
Tons per trip	40.96	
Tons per shift	1,231.98	
Average haul, miles	1.29	(Round trip)
Ton miles per shift	797.98	

When the mine was visited, the 18- by 25- by 24-ply tires on the new 40-ton trucks had not been in service long enough for service records to be developed. The 30-ton trucks formerly used were equipped with 16- by 25- by 20-ply tires. Thirty-six of these tires were driven 19,656 hours; 24 of them were then recapped and driven an additional 10,112 hours. This makes an average of 2,457 hours of use for a new tire and 1,264 hours for a recap. The average for a tire was about 26,399 miles or 3,299 hours including both new and recap duty. The truck duty at Copper Cities is about 8 miles per hour per operating shift.

One phase of the operation at Copper Cities that is considered utmost importance is preventative maintenance. This is particularly true of haulage equipment and is coordinated with daily operating service.

A grease shop is conveniently located near the crusher dump. This shop is operated by the mechanical department and is provided with two grease racks. Each truck is checked and greased once on each shift, and the check, by item, is indicated on a field report (fig. 23). Minor repairs are made, and those that cannot be made at the time are reported to the truck dispatcher. The truck dispatcher is a qualified mechanic who has experience in the mine operating department and reports directly to the mine superintendent. The dispatcher decides whether the equipment should be removed from service and sent to the shop or returned to service.

A record of the hours of operation for each truck is kept at the grease shop and at the mine superintendent's office. Special service functions, such as oil change, clean filters, transmission inspection, and rear-end service, are performed according to a regular schedule.

A service and fuel truck visits isolated equipment, and a charge slip is made and signed by both operator and garageman when petroleum products are issued (fig. 22). The operator of the supply truck completes a form to record all deliveries (fig. 24). The total charge is entered daily on an oil and gasoline record that covers each monthly period for each truck.

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COPPER CITIES DIVISION

MIAMI COPPER COMPANY
MINE DEPARTMENT

DUMP TRUCK DRIVER'S DAILY REPORT

TRUCK NO. _____

SHIFT _____ DATE _____

SHOVEL NO.	LEVEL	LOADS	
		ORE	WASTE

SPEEDOMETER READING-START _____
SPEEDOMETER READING-END _____
LOADS HAULED _____
TIME OPERATED _____
DELAYS - SERVICE _____
MECHANICAL _____
SHOVEL _____
_____
_____

GIVE OPERATING AND DELAY TIME IN MINUTES

_____
DRIVER

FIGURE 21. - Dump-Truck Driver's Daily Report.

Haul road construction has an important bearing on haulage costs. In 1957 the cost of constructing these roads amounted to 9.33 percent of the total haulage costs. The maintenance of clean levels has been pointed out; this aids materially in haul road construction. The road crew consists of a foreman and four bulldozer operators. Rooters and bulldozers are used to rough in a grade; then the surface is carefully smoothed with a grader (fig. 25). Roads are sprinkled with water continually while they are in use. The resulting surface is hard, smooth, and dustless. Dust preventative preparations are not used.

Auxiliary Operations and Services

Several auxiliary operations are carried on by Copper Cities mine organization. These and some of the necessary services are described in the following section.

Power

A 110-kv. power transmission line of the Salt River Power District crosses the property, and the voltage from this source is reduced to 6,900 volts by a 10,000-kv.-a. transformer at a substation near the south end of the concentrator. Power for the concentrator is delivered at 440 volts, and power for the mine is transmitted at 6,900 volts to a small transformer station at the mine where the voltage is reduced to 2,300.

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COPPER CITIES MINING COMPANY **Charge To Acct.** _____

Date _____ **Equipment No.** _____

DESCRIPTION	QUAN.	UNIT
DIESEL OIL		GAL.
GASOLINE—TAX PAID		GAL.
GASOLINE—TAX FREE		GAL.
LUB. OIL—KIND		GAL.
CHAIN OIL		GAL.
HOIST OIL		GAL.
CHASSIS GREASE		LB.
GEAR GREASE		LB.
ROLLER GREASE		LB.
GREASE—KIND		LB.
OTHER		

Garageman _____ **Operator** _____

FIGURE 22. - Fuel Charge Slip.

The high line to the mine is located so that the 2,300-volt transformer station can be placed near the center of the bench. Current is carried to the shovel by a four-strand insulated cable that is extended on radial lines and moved as the shovel changes position along the bank.

The cable is moved by one man and a pickup truck. Insulated tongs are fixed on one end of a short cable and the other end is fastened to the rear of the truck. Starting at the center near the transformer station, the man fixes the tongs to a 200- or 300-foot section of the power cable and pulls it to the desired location with the truck. This is repeated at locations farther from the transformer station until the power cable has been swung to the new radial position. At the bank end of the cable 200 or 300 feet of slack is retained, and this is played out as the shovel advances.

Water

Water is pumped from wells in the gravel along Miami Wash near Burch siding on the Southern Pacific railroad. This water is acid and contains a small amount of copper. After it is treated with lime, the sludge is pumped to the tailings area, and the slightly alkaline water is pumped from Burch to the 3,257,000-gallon reservoir above the mill by three multistage centrifugal pumps, each with a rated capacity of 750 gallons per minute (g.p.m.). Water

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TRUCK FIELD REPORT

FOREMAN _____ DATE _____

A
B
C

TR		DRIVER	MOTOR	BRAKES	FUEL SYSTEM	COOLING SYSTEM	OIL SYSTEM	TIRES	CLUTCH	TRANSMISSION	REAR END	HOIST	SPRINGS	SPINDLES	STEERING	CHS SPROCKET	SIDE BEAMS	TORQUE ARMS	FRAME	FRONT AXLE	REAR AXLE	BED	ELECT SYSTEM	ACCELERATOR	WHEELS	MISCELLANEOUS	B.O OR O.K.
61	A																										
	B																										
	C																										
62	A																										
	B																										
	C																										
65	A																										
	B																										
	C																										
66	A																										
	B																										
	C																										
67	A																										
	B																										
	C																										
68	A																										
	B																										
	C																										
69	A																										
	B																										
	C																										
63	A																										
	B																										
	C																										
64	A																										
	B																										
	C																										

FIGURE 23. - Truck Field Report.

is reclaimed from thickeners and from the tailings disposal area and added to this tank. This water is used for concentrating ore, for fire protection, and for sprinkling roads in the mine.

Drinking water is pumped from wells near the mine to a separate tank system with a capacity of 50,000 gallons.

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CASTLE DOME COPPER CO., Inc.
MINE DEPARTMENT

SERVICE AND FUEL TRUCK REPORT

SHIFT _____ DATE _____

ISSUED TO	FUEL OIL	GASOLINE	LUBRICATING OILS				CHAIN OIL	HOIST OIL	GREASE				
									ROLLER	GEAR 140			
TRUCK NO.													
NO.													
NO.													
NO.													
NO.													
NO.													
NO.													
NO.													
NO.													
NO.													
BULLDOZER NO.													
NO.													
NO.													
NO.													
NO.													
COMPRESSOR NO.													
NO.													
NO.													
NO.													
NO.													
NO.													
SHOVEL NO. 1													
SHOVEL NO. 4													
SHOVEL NO. 10													

DRIVER _____

FIGURE 24. - Service and Fuel-Truck Report.

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FIGURE 25. - Grading a Typical Haul Road.

Dump Leaching

At Copper Cities mine all waste material that contains oxide or sulfide copper minerals is placed in dumps so that the copper can be leached at some future date, but leaching is not done at the present time.

At Castle Dome, where mining has been completed, the mine dumps are treated with water and copper is recovered. Water is pumped on the Castle Dome mine dumps, allowed to percolate through them, and collected by dams in natural drainage channels as it leaves the dumps. Pumps return this pregnant water to a precipitation plant, where it is passed through concrete cells filled with detinned cans and the copper is precipitated. The sludge is picked up in a clamshell bucket with an overhead hoist and deposited in a hopper; it is then fed through a trommel, and the cement copper is separated from unused cans. The copper passes through openings of the trommel; it is washed, dried, and shipped to the smelter. The cans are then returned to the precipitation vats.

The water from which the copper has been precipitated is allowed to cascade down several banks into a sump (fig. 26). This oxidizes some of the iron, causing it to precipitate in the storage ponds, and thus lowers the soluble iron-salt content of the water before it is returned to the dumps. This results in decreased precipitation of iron salts and less cementing in the waste dumps. Water is recycled to the dumps through a 12-inch transite

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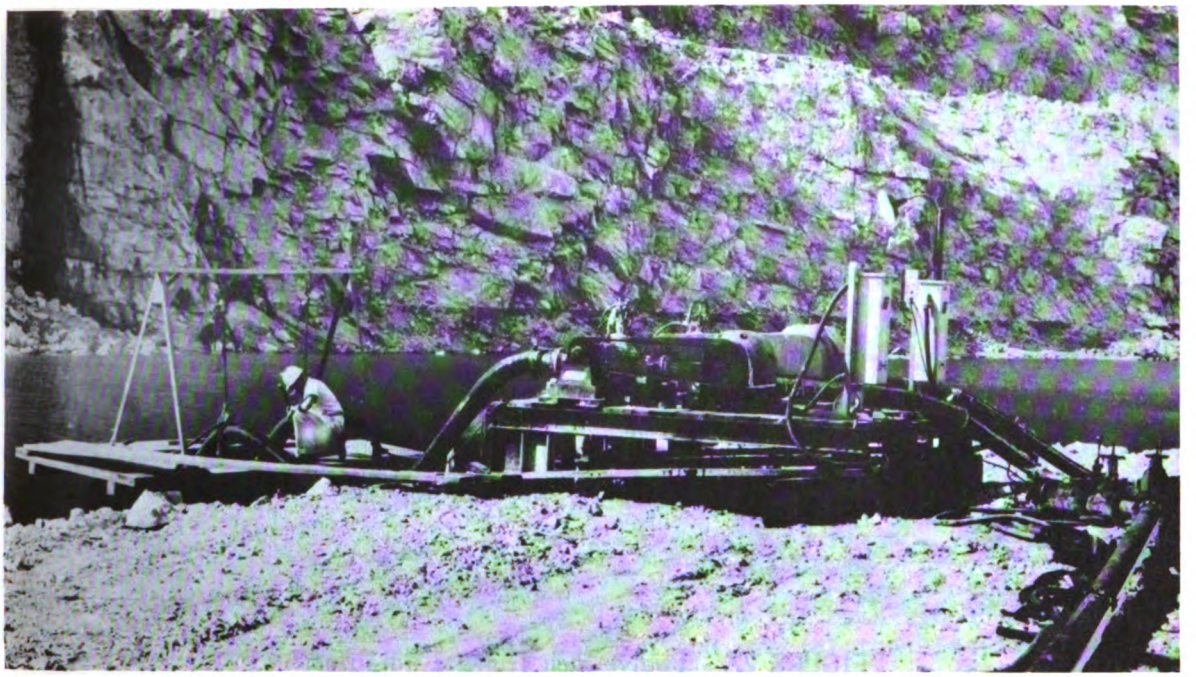

FIGURE 26. - Pump and Sump at Castle Dome Mine.

line (fig. 27) by a pump, which is mounted on a raft in the sump and connected to the transite pipe by a flexible hose.

A unique feature of this system is the method of distributing water on the dumps (fig. 28). Because water is not plentiful, operations can continue only by conservation of water in every possible way. Water loss from open ponds on the dump is quite high and the loss from sprays is excessive. This surface loss is avoided by driving a bull point about 6 feet into the dump and inserting a 1-inch plastic hose into the hole. The hose is inserted loosely and water permeates the dump, but if the hole becomes sealed there is no stoppage; the water backs up to the surface and flows along until it enters the dump.

The leaching-precipitating operation requires the employment of approximately 15 men per day; 4,233,462 pounds of copper was recovered in 1956. Solutions entering the treatment plant contain about 3 pounds of copper per ton of water.

Final Bank Slope

The Copper Cities mine pit is designed to have a final slope of 45°. Figure 29 is a plan of the banks that have been mined to the pit limit. The final slope at the Castle Dome mine, where the formations were similar, was approximately the same (fig. 30). These banks were observed in February 1957, about 5 years after mining was terminated. Some spalling had occurred, but the banks were still standing at approximately 45° (fig. 27).

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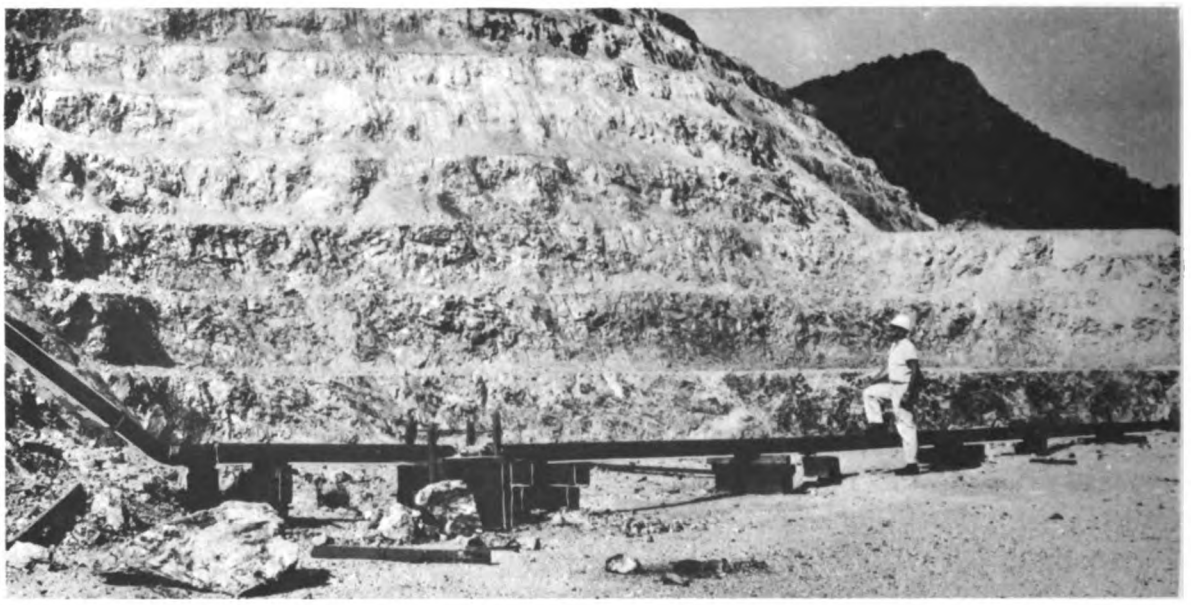

FIGURE 27. - Twelve-Inch Transite Waterline.

Waste Dump

Waste dumps are constructed level so leaching ponds can be constructed at some future date. A berm 3 to 4 feet high is built along the edge of the dump and along the haul road to the dump. This berm acts as a curb and prevents the heavy trucks from driving too close to the crest. As the dump subsides, a few more truckloads of waste are added and smoothed out to bring the elevation back to grade.

For night work, a tower has been mounted over a steel sled to support a floodlight. A 3-kw. generator powered by a gasoline engine supplies electricity. When it is necessary to move the tower the sled is hooked to a bulldozer and towed to the new location. A man is stationed at the dump to assist the driver spot the truck. When the shovel product changes from waste to ore, the man moves to the crusher dump.

Sampling and Ore Control

After the drilling and blasting foreman adjusts the location stakes for blast holes, they are numbered. As the hole is drilled, a wedge-shaped container is placed on the rim of the hole with the pointed end toward the rotating tool of the drill. Cuttings from the hole build up in a cone, and a segment of this cone falls into the container. The material in the container is split in a Jones-type splitter, and a 1- to 5-pound sample is sent to the assay office where it is analyzed for sulfide and oxide copper. Sample preparation in the drill area requires one man for about 2 hours per day.

As soon as the holes are drilled they are surveyed by stadia and their positions are plotted on a map of the mine. The mine map is drawn to a scale

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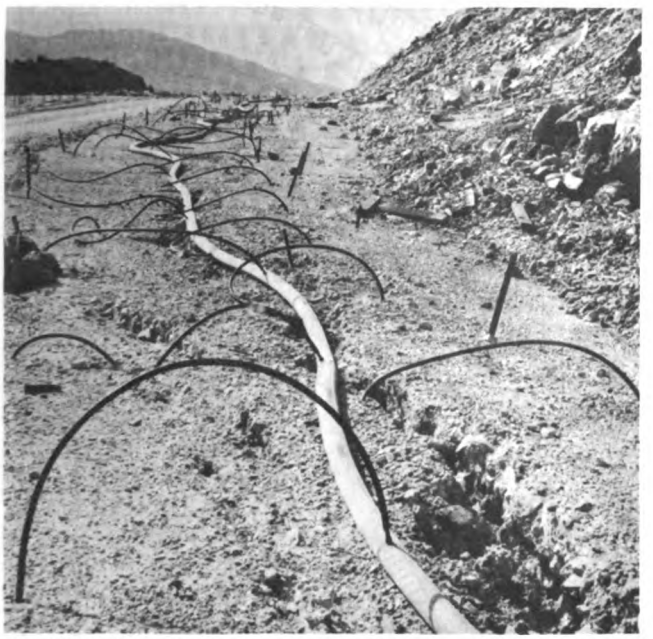

FIGURE 28. - Distributing Water for Leaching on Dumps.

of 50 feet to 1 inch, which allows room to record the assay figures near the location of the hole as the figures are received from the laboratory. After the bank has been blasted, targets are placed at ore-waste contacts by the engineer to assist the operating foreman locate ore or waste, as required.

MINE DRAINAGE

Tinhorn Wash formerly flowed across the area of the pit. A drainage ditch (fig. 5) was built to divert the flow to the east and around the pit. Some water seeps into the pit from springs along the back slope of the pit, and many of the blastholes are drilled in wet ground.

Drainage has been no problem except in blastholes, as all levels are open on the south side. As the mine becomes deeper, this will no longer be true, and pumps will be required. The amount of water will be small except during the torrential thunderstorms that occur during late summer months.

WAGE, CONTRACT, LEASING, AND BONUS SYSTEMS

Most employees are on an hourly wage scale. Supervisory employees are on a salary. The employees are represented by a union, and labor contracts are negotiated through the union. Contract, leasing, and bonus systems are not used.

A pension plan is provided for all employees. Upon retirement each employee who was on an hourly wage scale receives $2.25 per month for each year of continuous service up to a maximum of 35 years.

PERCENTAGE EXTRACTION AND PRODUCTION RATES

The mining system at Copper Cities mine provides for completing each working bench to the final pit limit. This is accomplished as rapidly as good mining practice allow. As soon as a level has been completed, a final report on that level is prepared by the mine superintendent. This report records and compares mined tonnage with the estimated tonnage. This comparison is illustrated in figure 31, and some important figures on extraction are given in table 5.

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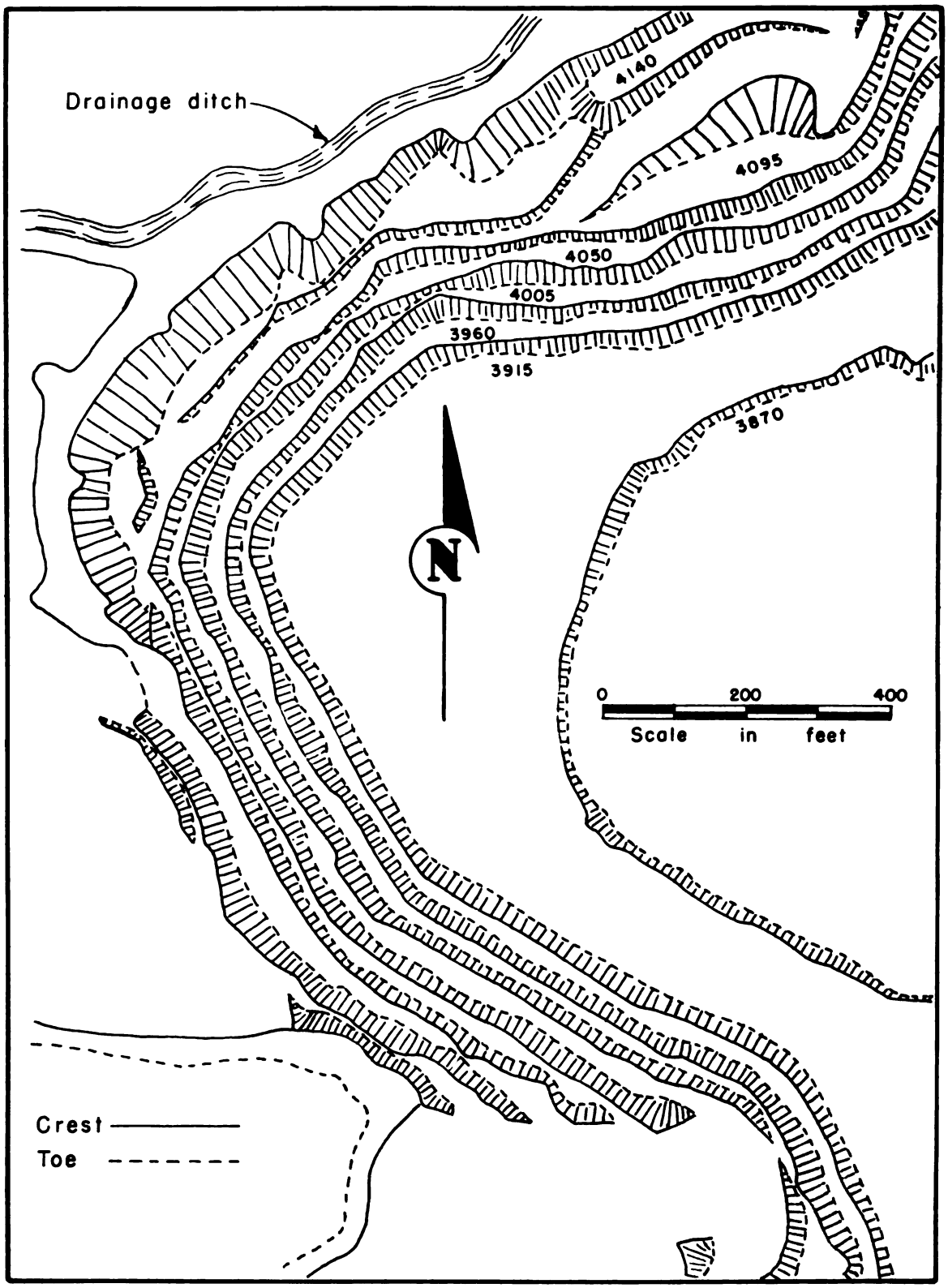

FIGURE 29. - Plan of Banks Trimmed to Trial Final Slope, Copper Cities Mine.

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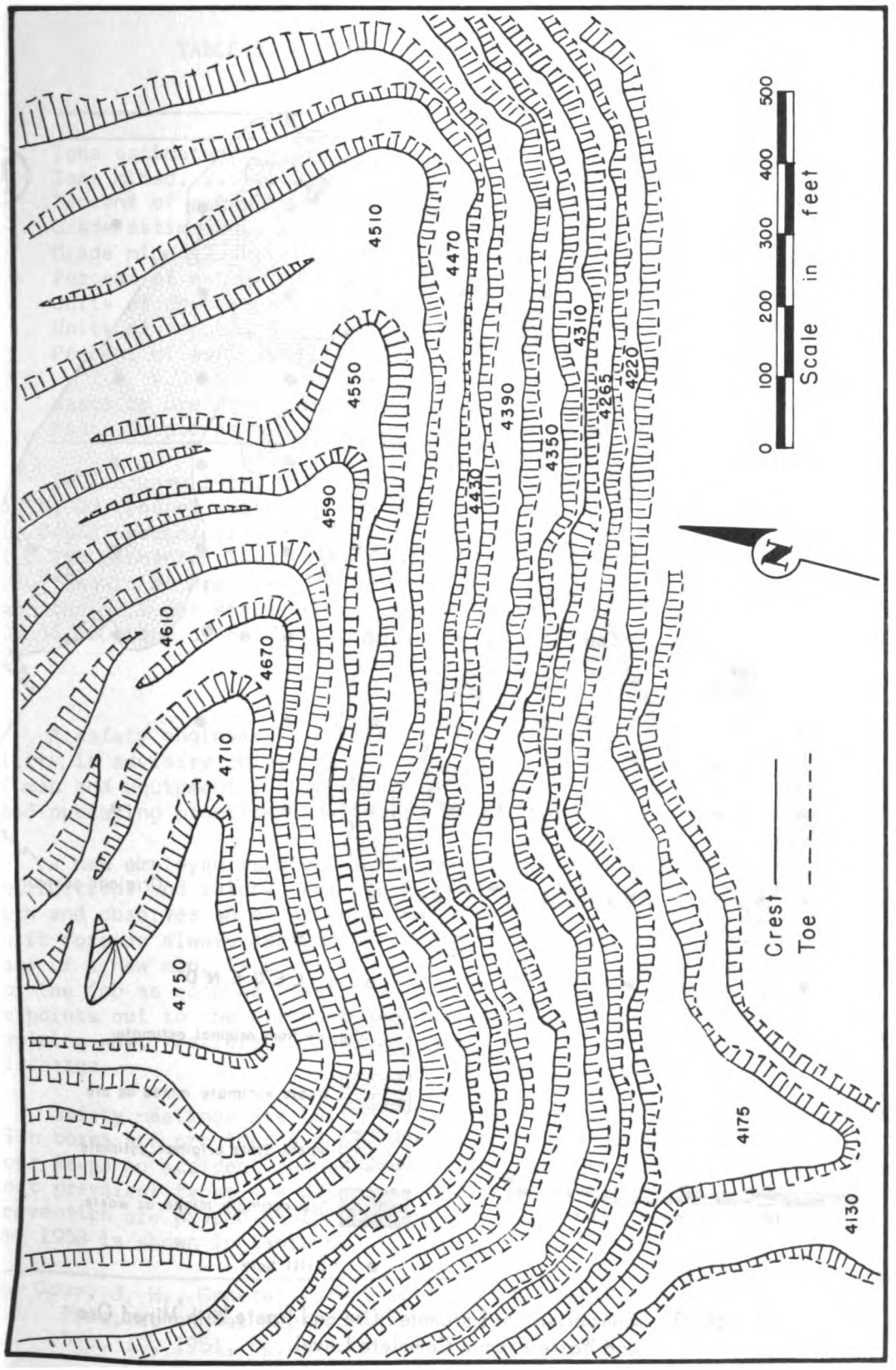

FIGURE 30. - Plan of Banks Trimmed to Final Slope, Castle Dome Mine.

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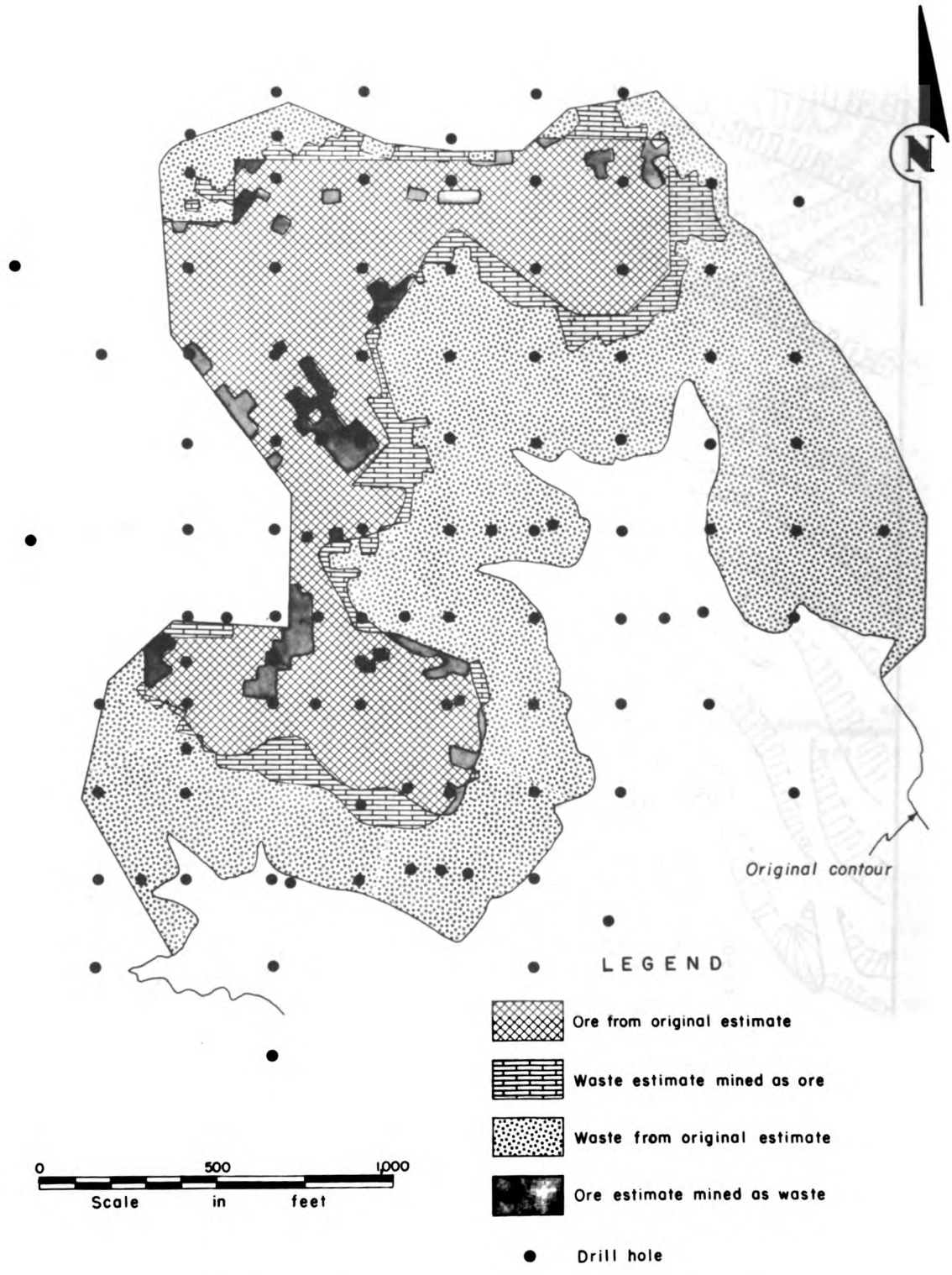

FIGURE 31. - Comparison of Estimated Ore and Waste With Mined Ore and Waste From a Complete Level.

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TABLE 5. - Final comparison for a completed level (tons estimated with tons mined)

	Ore	Waste
Tons estimated	4,165,000	6,358,280
Tons mined	4,332,535	6,372,801
Percent of estimate mined	104.02	100.228
Grade estimated	0.690	-
Grade mined	0.781	-
Percent of estimated grade mined	113.19	-
Units of copper estimated	2,873,850	-
Units of copper mined	3,383,524	-
Percent of estimated copper mined	117.73	-
Waste to ore as estimated	1.53 - 1.00	-
Waste to ore as mined	1.47 - 1.00	-

In February 1958, the mine was producing approximately 12,000 tons of ore and 8,000 tons of waste per operating day. During 1956, the mine produced 55,264,337 pounds of copper from 4,167,147 tons of ore, with a copper content of 0.795 percent. That year, 3,869,132 tons of overburden was stripped, resulting in an ore-to-waste ratio of 0.93 to 1.00. Between August 2, 1954, when the mill was started, and December 31, 1957, the mine had produced 12,566,000 tons of ore. About 31,613,000 tons of waste had been stripped.

SAFETY

A safety engineer is in charge of mine and plant safety. His responsibility is advisory only; the operating foreman is responsible for the safety of men and equipment in his charge. Safety is considered an integral part of good operating practice and therefore is a supervisory function.[12/]

A new employee is given a thorough physical examination and a book of instructions and rules. For the first shift or longer, if necessary, he works with and observes an experienced man on a job that he will later assume. The shift foreman always has ample time to observe each man everyday and, in the case of a new man, is particularly instructed to determine the man's fitness for the job as soon as possible. Each time a foreman observes bad practices, he points out to the offender the dangers involved to life and equipment, and explains why. Failure to observe safety rules is a valid cause for summary discharge.

Safety meetings are held by the mine superintendent for foremen. Suggestion boxes are provided and all suggestions are considered. The safety engineer keeps an accident record and summarizes the accidents by their type. The most prevalent types are discussed at safety meetings and suggestions for their prevention are passed on to the men. The accident record for the years 1957 and 1958 is shown in table 6.

12/ Gray, J. H., General safety to men at Castle Dome and Copper Cities Pits: Pres. at annual meeting of Ariz. and Sec., AIME, Tucson, Ariz., Dec. 10, 1951, pp.

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TABLE 6. - Accident record, Copper Cities Division

Department	Number of Accidents	Man-hours worked	Frequency	Days lost	Severity
1957:					
Mine.....	2	194,783	10.268	61	0.313
Mill.....	10	349,796	28.588	102	.292
Surface.....	13	422,187	30.792	108	.256
Total.....	25	966,766	25.859	271	0.280
1958:					
Mine.....	1	154,814	6.459	14	0.090
Mill.....	1	286,959	3.485	25	.087
Surface.....	4	323,026	12.348	180	.556
Total.....	6	756,699	7.836	219	0.286

FIRST AID

Good roads lead to all points of employment in the mine and the plant. Miami Copper Co. in cooperation with Inspiration Consolidated Copper Co. operates a hospital and outpatient clinic, and an ambulance is available to Copper Cities mine at all times. The primary objective of first aid is to get an injured employee to the doctor as soon as possible. First-aid classes are scheduled for employees only when they will serve a useful purpose.

FIRE PREVENTION

The 3,257,000-gallon water reservoir for the mill is situated above all buildings and water for fire hydrants is fed by this supply. Extinguishers of a suitable type are placed near electric motors and other special hazards. Equipment is provided at company expense and a fire-fighting crew is trained to answer a whistle signal.

MINE ORGANIZATION

The operating mine organization consists of about 68 employees on hourly pay and 13 employees on salary. The distribution of these employees is shown in figure 32. The activities of the mine are summarized daily by the mine foreman on a form (fig. 33).

PROCESSING PLANT

A complete description of the concentrating plant is beyond the scope of this paper. The plant equipment at the Copper Cities mine is essentially the same as that used at the Castle Dome mine. That mill began operating on June 10, 1943, and ceased operations at the Castle Dome location on December 5, 1953; then it was moved 15 miles to the Copper Cities mine, where it began operating on August 2, 1954.[13] Table 7 presents a summary of the Castle Dome and Copper Cities operations.

13/ Henderson, T. C., Copper Cities Concentrator: Pres. AIME, Miami, Ariz., June 1, 1957, 7 pp.

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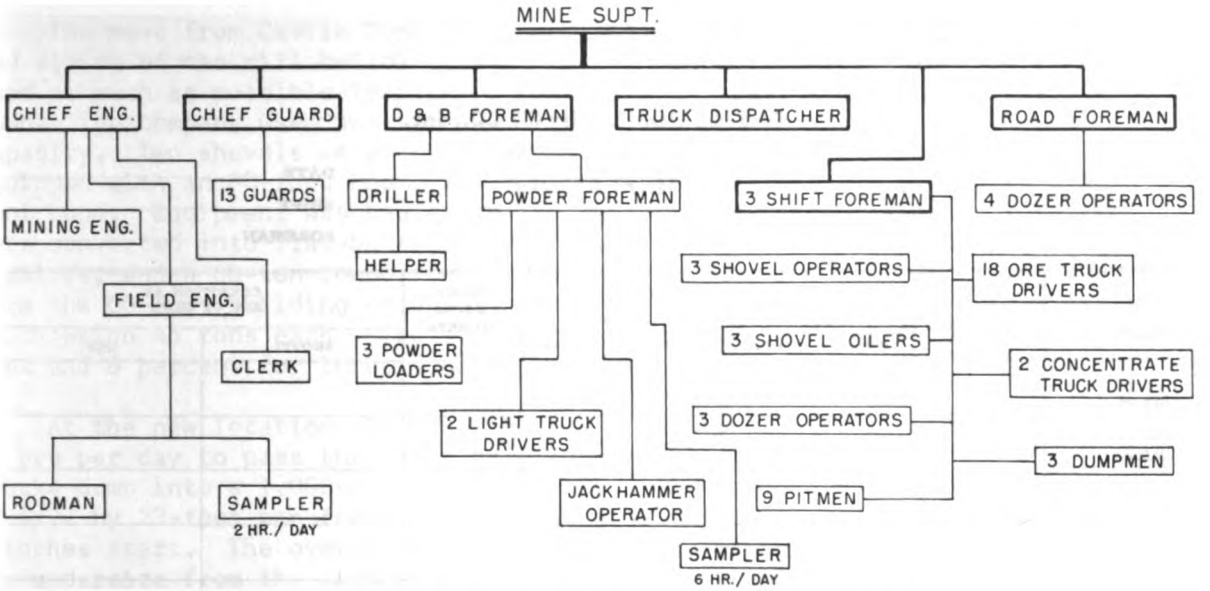

FIGURE 32. - Mine Organization Chart, Copper Cities Division.

TABLE 7. - Summary of concentrator operations

	Castle Dome mine	Copper Cities mine[1]
General:		
Mill started	June 10, 1943	Aug. 2, 1954
Mill closed	Dec. 5, 1953	-
Tons milled	41,442,616	12,566,000
Grade of ore, percent:		
Total Copper	0.725	0.715
Sulfide	0.689	0.680
Oxide	0.036	0.035
Tons milled per man shift	71.7	57.6
Water recovery.....percent	81.7	86.6
Reagent and steel consumption, pounds per ton:		
Sodium Aerofloat	0.050	0.050
Pine oil	0.126	0.087
Dowfroth	-	0.030
Lime	3.60	3.69
Grinding balls	1.92	1.58
Ball mill liners:		
Manganese liners	0.012	-
NiHard liners	0.099	-
Cast iron liners	0.007	-
Power consumption; kw.-hr. per ton	14.283	11.952
Water consumption; gallons per ton of ore:		
Reclaimed	635.0	-
Fresh	116.0	130.0
Copper recovery, pounds per ton of ore	12.408	11.906

1/ Figures cover period from Aug. 2, 1954, to Jan. 1, 1958.

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CASTLE DOME COPPER CO., Inc.

MINE DEPARTMENT

SHIFT REPORT

DATE .....

SHIFT .....

FOREMAN .....

SHOVEL	SHOVEL HOURS WORKED	TRUCK LOADS DUMPED				TRUCK HOURS WORKED	CONDITION AT END OF SHIFT	
		ORE		WASTE				
		30 Ton	14 Ton	30 Ton	14 Ton		SHOVEL	PIT
M5006								
M5007								
M5008								
M5053								
TOTAL								

MEN WORKED	
DUMP BOSS	
SHOVEL OPERATORS	
SHOVEL OILERS	
DOZER OPERATORS	
AUTO CAR DRIVERS	
TRUCK DRIVERS	
SPRINKLER DRIVER	
PITMEN	
OTHER	
TOTAL WORKED	
MEN OFF	
VACATION	
LEAVE	
SICK	
INJURED	
A. W. O. L.	
OTHER	
TOTAL OFF	
TOTAL MEN ON SHIFT	

GENERAL REMARKS:

Accidents, shovel breakdowns, repairs needed, condition of roads, dump, pit, shovel moves, etc.

FIGURE 33. - Shift Report.

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The move from Castle Dome to Copper Cities, except for structural steel and siding of the mill building, was done by company employees. Cranes were used as much as possible to pick up machinery before the buildings were torn down. The company used two rubber-tired mobile cranes each with a 15-ton capacity. Two shovels were converted into cranes, a 1-1/2-yard shovel was equipped with an 80-foot boom, and a 5/8-yard shovel was equipped with a 50-foot boom. Equipment was hauled in two 18-foot ore-concentrate trailers that were converted into flat-bottomed trailors, four smaller flat-bottomed trailors, and a 60-ton lowboy unit that was rented. The longest items moved were the 66-foot building columns and the heaviest pieces were the ball mills, which weigh 45 tons each. The steepest grades were 12 percent for about 1,000 feet and 8 percent for three-fourths of a mile.

At the new location the crushing plant was rebuilt to reduce 12,000 tons of ore per day to pass through a 5/16-inch mesh screen. The 40-ton mine trucks dump into a 1,000-ton receiving bin. The bottom of this bin is a 10-1/2- by 23-foot pan feeder that feeds a grizzly with sloping bars spaced 8 inches apart. The oversize goes directly into a 66- by 84-inch jaw crusher. The undersize from the crusher and the grizzly is conveyed to a second-stage crushing plant composed of two 7-foot standard cone crushers in an open circuit, and then to a third-stage crushing plant where four shorthead cones are used in a closed circuit with 5/16-inch screens to size the final product. A simplified flowsheet of the crushing plant is shown in figure 34. Important equipment is as follows:

Crusher equipment

1. Ore pocket (1,000 tons).
2. Pan feeder (10-1/2 by 23 feet).
3. Grizzly (8-inch spacing).
4. Jaw crusher (66 by 84 inches).
5. Conveyor (4 by 161 feet).
6. Grizzly (4-inch spacing).
7. Conveyor (4 by 523 feet).
8. Storage bin (578 tons).
9. Pan feeder (4 feet by 16 feet 11 inches).
10. Grizzly (1-1/4-inch spacing).
11. Standard cone crusher (7 feet)
12. Conveyor (4 by 1,054 feet).
13. Storage bin (800 tons).
14. Conveyor (3-1/2 by 215 feet).
15. Vibrating screen (5/16- by 1-inch mesh).
16. Vibrating screen (1/2- by 1-inch mesh).
17. Conveyor (3-1/2 by 43 feet).
18. Shorthead cone crusher (7 feet).
19. Conveyor (3 by 63 feet).
20. Conveyor (3 by 240 feet)
21. Conveyor (3-1/2 by 254 feet).
22. Conveyor (3-1/2 by 1,181 feet).
23. Fine-ore bin (12,000 tons).

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The final product of the crushing plant is stored in a 12,000-ton suspension-type steel bin. Feeder chutes from this bin discharge on a belt system that feeds over a weightometer to the ball mills. The mills are charged with 2- and 3-inch balls and driven at 16.6 r.p.m., or about 68 percent of their critical speed.

For flotation six 100-foot primary cells of Miami type are used. About 4 pounds of lime is used to neutralize the acid, 0.017 pound of Dowfroth and 0.061 pound of pine oil are used as a frothing agent, and 0.049 pound of sodium Aeroflotat is used as a collector. Concentrates are cleaned and recleaned in 35-foot Miami-type cells, and rejects are passed through 10-inch cyclones; underflow is returned to the ball mill. Concentrates are filtered, hauled in 20-ton end-dump trailers 3 miles to a railroad siding, dumped in a receiving bin, and loaded into railroad cars by conveyors; the cars are then taken to the Inspiration Smelter. A simplified flowsheet of the grinding and floation is shown in figure 35; important equipment is as follows:

Grinding and flotation equipment

1. Fine ore bin (12,000 tons).
2. Ore feeders (5 used).
3. Conveyor (2 by 84 feet).
4. Conveyor and scales (2 by 75 feet).
5. Automatic pH control units.
6. Ball mill (10-1/2- by 11-foot grate type, with drum feeder and 600-hp. motor).
7. Classifier (12- by 28-foot rake type).
8. Six-way distributor.
9. Primary flotation (six Miami-type air lift cells, each consists of two 50-foot sections).
10. Cyclone separator (10 inches).
11. Cleaner flotation (35-foot Miami-type cell).
12. Regrind ball mill (7 by 12 feet, with 250-hp. motor).
13. Thickener (60-foot diameter).
14. Leaf filler (four 8-1/2- by 6-foot leaves).
15. Conveyor (1-1/2 by 316 feet).
16. Conveyor (1-1/2 by 46 feet).
17. Concentrate tailer (20 tons).
18. Thickeners (225-foot diameter).
19. Hydroseparator (30-foot diameter).
20. Tailings dam.

Typical properties of the products of the Copper Cities mine are shown in table 8.

Final tailings from the rougher cells flow through a rubber-lined wood launder to two 225-foot thickeners. Thickened tailings that average about 45.2 percent solids flow by gravity through 18-inch redwood stave or transite pipes on an 0.8 percent grade. Tailings are discharged into the pond from 4-inch spigots that are spaced at 39-foot intervals.

Water from the thickeners is returned to the mill supply tank. At times slimes are troublesome, and a flocculant (Separan))is added in amounts of about 0.003 pound per ton of ore. Phosphate is added in amounts of 1 to 2 p.p.m. at several key places where scaling and choking pipelines have occured.

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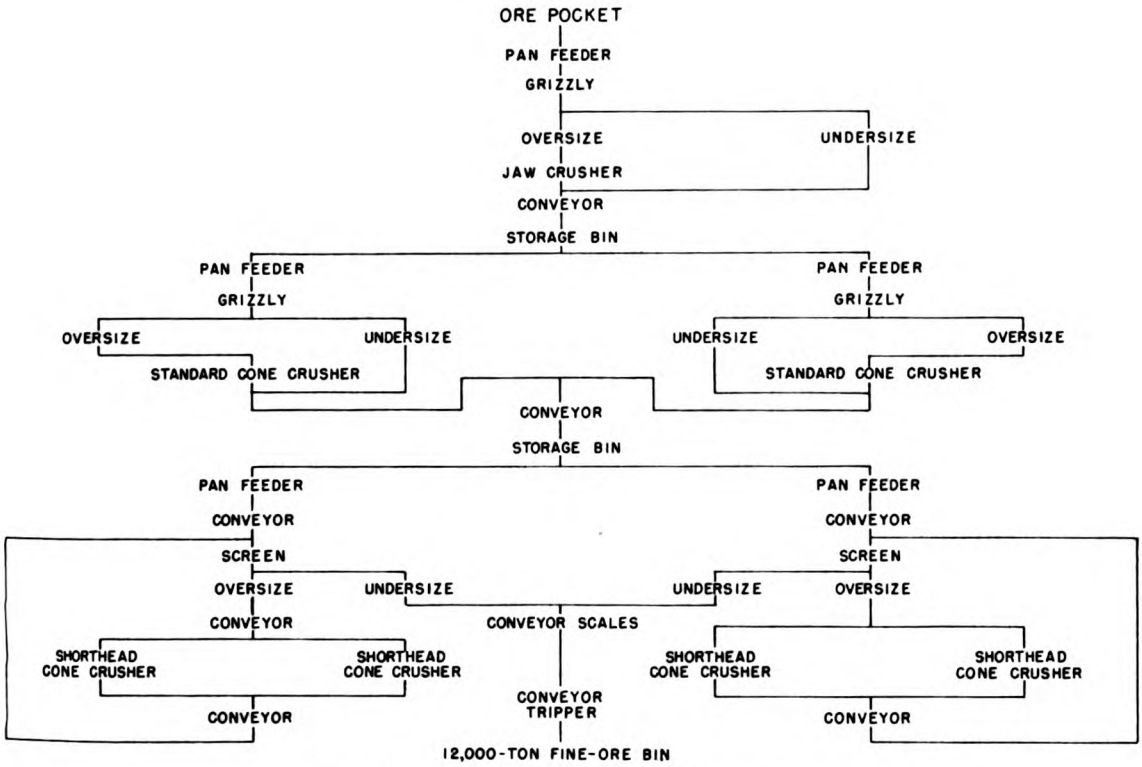

FIGURE 34. - Crushing-Plant Flowsheet.

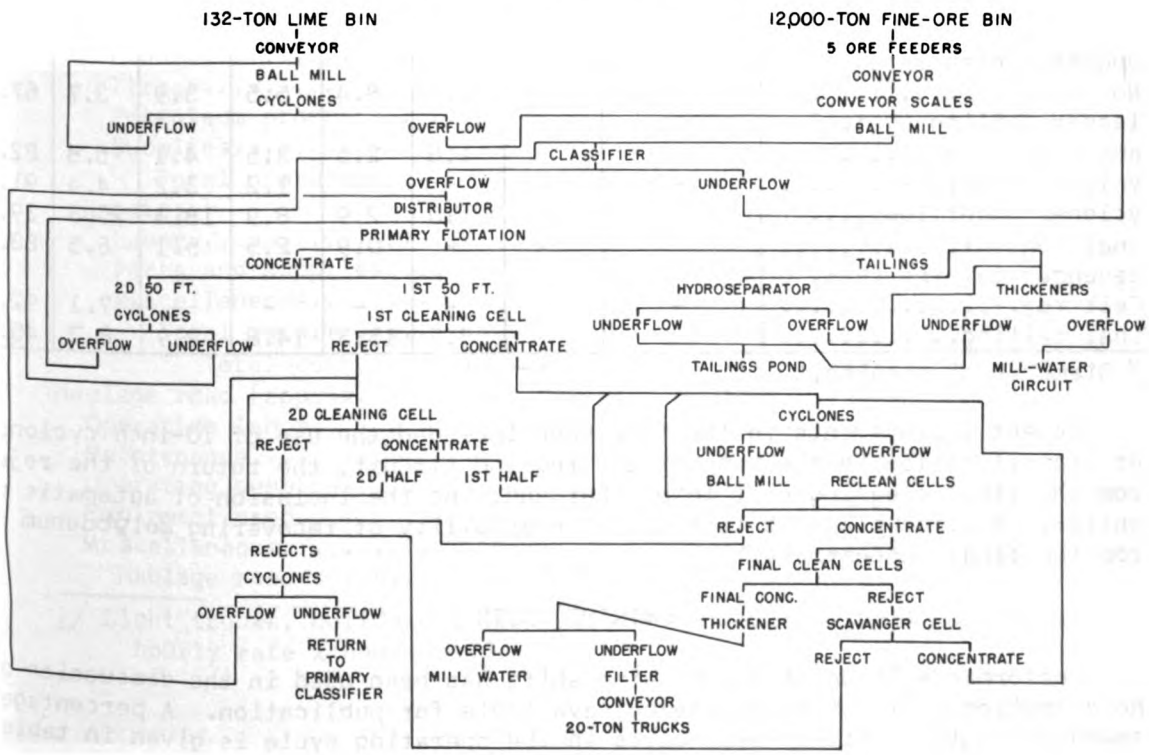

FIGURE 35. - Grinding and Flotation Flowsheet.

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TABLE 8. - Typical properties of Copper Cities products

Product	Solids, average percent	Free CaO, pounds per ton of solution	Average assay percent			
			Total Cu	Sulfide Cu	Fe	Insol.
Crusher product.....	97.1	---	--	--	--	-
Flotation feed.....	24.3	0.047	0.857	0.815	2.0	89.8
Rougher concentrate No. 1.....	7.0	-	20.160	-	21.7	24.2
Rougher concentrate No. 2.....	3.1	-	2.230	-	9.6	73.4
Cleaner tailing (Ret. 3 and 5 Cell rej.).....	2.1	-	1.960	-	24.8	40.6
Cyclones overflow.....	15.4	.103	22.600	-	27.6	13.2
Cyclones underflow.....	81.7	-	15.480	-	35.8	5.0
Final concentrates.....	24.9	-	38.950	37.650	20.6	6.8
Scavenger tailing (Ret. 6 Cell rej.).....	2.9	-	2.920	2.670	26.3	37.8
Final tailing.....	23.4	.046	.113	.066	1.3	93.8
Filtered concentrate.....	91.2	-	-	-	-	-

	Average screen analysis, percent on mesh							
	+0.371[1]	+35	+48	+65	+100	+150	+200	-200
Crusher product.....	15.3	68.8	2.5	2.0	1.9	1.4	1.3	6.8
Flotation feed.....	-	-	7.0	14.4	16.2	10.1	9.0	43.3
Rougher concentrate No. 1.....	-	-	4.3	9.5	16.7	14.4	12.2	42.9
Rougher concentrate No. 2.....	-	-	11.3	8.4	5.5	3.9	3.7	67.2
Cleaner tailing (Ret. 3 and 5 Cell rej.).....	-	-	1.8	2.5	3.5	4.1	5.8	82.3
Cyclones overflow.....	-	-	-	-	1.2	3.2	4.4	91.2
Cyclones underflow.....	-	-	1.0	2.9	8.9	18.4	29.3	39.5
Final concentrates.....	-	-	-	0.9	2.5	5.1	8.5	83.9
Scavenger tailing (Ret. 6 Cell rej.).....	-	-	-	-	-	-	7.1	92.9
Final tailing.....	-	-	8.2	13.3	14.8	9.9	8.7	45.1

1/ Diameter of opening.

Recent improvements in the flow have included the use of 10-inch cyclones for classification in the concentrate regrind circuit, the return of the reject from the final cleaning cell to be reground, and the inclusion of automatic pH control. A pilot plant is testing the possibility of recovering molybdenum from the final concentrate.

MINING COSTS

Performance in units per machine shift has been used in the discussion of the operation. Direct costs are not available for publication. A percentage breakdown of the costs of some units in the operating cycle is given in table table 9.

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TABLE 9. - Mine equipment unit costs

	Percentage breakdown	
Rotary drill:		
Operation:		
Labor	30.56	
Supplies	20.17	
Total operating		50.73
Repair:		
Labor	26.39	
Parts and supplies	8.16	
Miscellaneous	14.72	
Total repair		49.27
Total cost		100.00
Loading, 5-yard shovel:		
Operating:		
Labor	27.17	
Supplies	2.87	
Power	2.63	
Miscellaneous	28.14	
Total operating		60.81
Rapair:		
Labor	16.03	
Parts	23.16	
Total repair		39.19
Total cost		100.00
Haulage, 40-ton trucks:		
Operating:		
Labor	39.50	
Tires	18.48	
Petroleum products	11.16	
Supplies	2.71	
Total operating		71.85
Repair:		
Labor	18.05	
Parts and supplies	8.67	
Miscellaneous	1.43	
Total repair		28.15
Total cost		100.00
Haulage road (approx. 7 percent of total haulage cost):		
Operating labor	23.08	
Maintenance	1.23	
Operating supplies	4.10	
Equipment used	1/69.51	
Miscellaneous	2.08	
Haulage road costs		100.00

1/ Light trucks, bulldozer, carryall, and other equipment charged at hourly rate x hours used.

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